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**The Patent Situation.**

The legal importance of the much discussed U. S. Supreme Court decision in the Dick Co. case rests in the fact that it authoritatively fixes the extent of the monopoly guaranteed by a patent as absolute. A patent monopoly is guaranteed by the Constitution, but the extent of this monopoly, not being fixed by statute law, has been somewhat indefinite in the past, as is shown by the discrepancy between different lower-court decisions. Now this uncertainty is ended.

There is, however, a very practical side to this question. While reserving a discussion of this for a future occasion, this one point we want to make here, namely that in most discussions of the patent situation, two things are usually mixed up—the patent law and its working in practice or enforcement. We believe that the American patent law as law is pretty good and perhaps the best in the world, but that its practical operation, its enforcement, is about as poor as it can be, as under existing conditions the guaranteed protection can be enforced by an individual inventor only if he is backed by immense wealth.

The patent situation will be a national issue in a very few years. As its solution will have to be found by a "small representative part" of the whole people, engineers should make up their mind now that they are rightfully among those most concerned and should be represented. There is a hopeful awakening to the seriousness of the situation especially in chemical circles. When reasonable people begin to think over a thing and make up their mind what they want, they generally get it.

**Concentration by Flotation.**

Possibilities for improvement in concentration are to be found in the use of machines and methods which are only accessory to the well recognized and accepted practice of the day. Processes which are heralded as "revolutionary" are apt to receive scant consideration, but when a new proposal is made and its possible field of usefulness carefully outlined it is likely to be investigated. The flotation of dry crushed ore, described elsewhere in this issue, is such an accessory process, and doubtless will find special application in conjunction with standard methods of concentration.

The tendency of some sulphide minerals to float, unassisted by mixture with oil and acid, or the use of a vacuum, is not a newly discovered phenomenon. The relation of these substances to the surface tension of water is such that particles as large as eight or ten-mesh in size, which have been freshly crushed in the dry state, may be deposited on the surface of water without overcoming the tension of its surficial film. Indeed wetted particles of minerals may be exposed to the air and then delivered onto the surface of water without sinking. This latter phenomenon has been noticed many times on con-

centrating tables when treating certain minerals which show strongly the tendency to float. Such mineral usually is lost, and in some extreme cases it has been found useless to attempt concentration because the loss due to floating mineral was prohibitive.

We may consider, for example, the concentration of tetrahedrite carrying silver. This mineral is exceedingly friable, and the loss in ordinary concentration is sometimes very great. Resort may be had to the best systems of slime concentration, but even then the recovery is very imperfect. On the other hand, tetrahedrite shows a marked tendency to float on water, and so a way is opened to improve concentration by flotation. The freshly crushed ore may be treated by flotation first, and the fine floating concentrate recovered before it enters the mill system. Thus that portion of the ore which ordinarily is most difficult to recover, is at once saved, and the balance may be treated in the usual manner. An instance recently came to our notice where the use of flotation as an accessory to the regular practice permitted the profitable operation of a mine and mill which hitherto had been unprofitable.

#### Publicity in Nevada.

Perhaps no mining state has had more intimate experience with booms than Nevada. The Comstock days are only a matter of history to many of us, but the more recent excitement caused by the discovery of Goldfield is still fresh in the mind of anyone familiar with the mining industry. Mining promotion during the Goldfield boom required only a vivid imagination, a good press agent and the usual supply of printed and engraved paper. Fortified with these, the fiscal agent opened an office in an eastern city and employed an officer to keep the people in line while they besieged his office for the purpose of buying stock. The ownership of a mining claim, or the expenditure of real money in development was not considered necessary by the wild-cat companies which did Nevada so much harm during the Goldfield boom. It has been estimated that \$30,000,000 would represent a conservative estimate of the cost of that boom.

Having suffered this affliction, Nevada determined on a cure which would be effective for all time, and the result is that mining and milling companies operating in that state are obliged to make public the important details of their work. The state levies a bullion tax on the output of all mines, and the amount is determined from the sworn statements of operation made by company officers. These statements, which include a report of tonnage mined and treated, its value and the expense incurred in those operations, are made public by the state bullion tax collector, so that all concerned in the property may know whether it is in operation and to what extent. The last statement issued contains authentic data on sixty-one companies and forms an admirable publicity report. There is one desirable thing the state cannot do, however, and that is to compel the public to read the statement and inform itself before investing its money.

#### Motion Study in Mill and Laboratory.

Inquiry among western technical men reveals a commendable tendency to apply the principles of scientific management to metallurgical works and laboratories. This kind of investiga-

tion has been developed most highly, and with gratifying results, in eastern industrial plants; but there is reason to believe that the idea has taken root in the field of metallurgy and will continue to develop until that branch of scientific industry has been greatly benefited.

Quite naturally the idea is being applied first to those operations which are routine in their nature, consisting of a number of separate motions or operations repeated many times throughout the period of labor. Shoveling, tramping and wheeling come under this head, as also do the operations of loading and dumping cars on aerial tramways. The different elements of such work can be plotted and presented to the mind graphically, and by studying the results it will be possible to see where time can be saved or capacity increased without hardship to the laborer and possibly to his advantage. Standards can thus be established to serve as a basis of labor and wage, and adjustments can be made whereby greater efficiency will gain extra remuneration.

Some of the routine work of the laboratory also can be standardized to the great advantage of the chemist. The arrangement of the laboratory and the use of appropriate apparatus are important factors in reducing the number of motions or steps necessary to do a day's work. A standard of efficiency can be established, based on the accuracy of the work as gauged by the number of determinations necessary to be repeated, or the percentage of umpires won and lost. In this way a healthy rivalry can be produced and the standard of efficiency kept high.

In order successfully to carry out these ideas there must be broadminded management, definite organization and perfect co-operation. The benefit of the laborer must be kept in mind in order that the company ultimately may reap the results of his interest and improved work. As yet the application of the idea is limited in metallurgical work, but those who have endeavored to apply it are enthusiastic over the results obtained and see great possibilities in further application. We join them in urging upon technical men everywhere a general study of the principles of scientific management and the publication of the results obtained in order that the whole industry may prosper and make progress.

#### The Steel Corporation's Report.

The United States Steel Corporation's annual report, issued under date of March 19, 1912, and covering the calendar year 1911, contains as usual a full statement of the corporation's operations in the year. An approximate statement of the year's earnings has been available since the publication of the statement for the fourth quarter, but the final figures, as usual, contain the result of various minor adjustments made after the close of the year.

The corporation began business April 1, 1901, its earnings in the remaining nine months of that year being \$84,787,597, but no statement of production in the period was made. For subsequent years we present below a table showing (1) total earnings, being the earnings after provision for ordinary maintenance and repairs and payment of interest on subsidiary company bonds, but before allowances are made for sinking funds against subsidiary company bonds, or the regular allowance for depreciation, interest and sinking funds against the

steel corporation bonds, etc.; (2) the steel products for sale, comprising material in the form in which it passed to customers of the steel corporation, these totals, according to the practice of the steel corporation, being made up of gross tons for all products, except that wire and products of wire and finished structural work are represented in net tons; (3) our computation of the average earnings per ton:

	Total earnings.	Steel products for sale	Average per ton.
1902.....	133,308,764	8,033,556	\$16.59
1903.....	109,171,152	7,458,879	14.64
1904.....	73,176,522	6,792,780	10.77
1905.....	119,787,658	9,226,386	12.98
1906.....	156,624,273	10,578,433	14.81
1907.....	160,964,674	10,376,742	15.51
1908.....	91,847,711	6,206,932	14.80
1909.....	131,491,414	9,859,660	13.34
1910.....	141,054,755	10,733,995	13.14
1911.....	104,305,406	9,476,248	11.07

What is particularly impressive about the Steel Corporation's operations is that the average earnings per ton vary less than would be expected from observation of the vicissitudes in the trade. When prices are high and mills are operating full and therefore presumably at greatest efficiency the earnings per ton do not rise, and when prices are low and production is far below capacity the earnings per ton do not fall as much as would be expected. For example, the years 1906 and 1907 were years of full operation, except for a sharp decrease in November and December of 1907, and earnings averaged \$15.16 per ton in the two years. In 1911 prices were \$5 to \$7 per ton lower and production 10 per cent less, yet earnings averaged only \$3 per ton less.

It must be remembered, of course, that what are loosely called the "high fixed charges" of the Steel Corporation are quite largely charges which are paid after these "total earnings" are ascertained, and the factor of decreased operations affects the balance of earnings available for common stock dividends much more than it does the so-called "total earnings." The corporation's fixed expense, paid before the "total earnings" are reported, amounted in 1911 to about \$34,000,000, this being made up of somewhat more than \$16,000,000 of administrative and other general expense, a trifle less than \$10,000,000 of taxes and about \$8,000,000 interest on obligations of subsidiary companies. These items vary but little from year to year, whether production is high or low, varying well inside the limits of \$3 and \$6 per ton in the years of maximum and minimum tonnage respectively.

After meeting the requirements of the depreciation and extraordinary replacement funds, the bond sinking funds and paying interest on bonds, 7 per cent on the preferred stock and 5 per cent on the common stock, there was left as the 1911 contribution to surplus the sum of \$4,665,495, which is less than 50 cents per ton of the 1911 production. During the year there was a decline in the average selling prices of finished steel products, except rails, of about \$5 per ton, so that obviously the Corporation was very close to the line in the matter of earning its regular dividends on the common stock. This dividend amounted to about \$2.70 per ton of output.

The Corporation to-day is operating to better advantage than in 1911. Prices are lower, it is true. They are not an average of \$1 per ton higher than the minimum level of 1911, and they are nearly \$2 a ton lower than the average level of that year.

On the other hand, production is at a much better gait. The United States as a whole is making pig iron at the rate of about 27,000,000 tons per annum, the actual production in 1911 having been only 23,649,344 tons, while the Steel Corporation is operating at between 85 and 90 per cent of its full capacity against an average of only 67 per cent in 1911, as we learn from a statement in the annual report.

The Steel Corporation's exports in 1911 of rolled steel and other finished products amounted to 1,719,272 tons. The country's total exports of all such products amounted to 1,955,924 tons, indicating that the corporation conducted 88 per cent of the export business in these lines. Its exports of pig iron and scrap amounted to 26,728 tons, the total exports of the country being 108,717 tons. There were also exports of bar iron and of cast radiators and house heating boilers, essentially tonnage products of the iron and steel industry, to which the corporation did not contribute.

The operations at the Gary plant were of particular interest. Production of the Indiana Steel Company in 1911 was as follows, in tons:

By-product coke.....	811,804
Pig iron.....	707,273
Steel ingots.....	1,036,545
Rails .....	281,980
Other rolled steel.....	469,366
Total rolled steel.....	751,340

There are eight blast furnaces at Gary, rated at 1,200,000 tons per annum, the 1911 output thus being 59 per cent of the rated capacity. While the first battery of by-product ovens was not put in operation until April 12, 1911, four other batteries coming in at later dates in the year, it is probable that the by-product plant made nearly all the coke required by the blast furnaces in 1911, blast furnace operations at Gary having been light early in the year. The total output of rolled steel only slightly exceeded the production of pig iron, and it may be inferred that little outside scrap was used. The production of rolled steel was 72.5 per cent and the production of pig iron 68.2 per cent of the steel ingot production, but of course these comparisons throw no light upon the average proportions of scrap and pig iron charged into the open-hearth furnaces, even an approximation to the proportion being impossible from these data.

The report refers to the cancellation of the lease of the so-called James J. Hill or Great Northern ore properties, but makes no effort to suggest the reason which led to the cancellation. It is merely stated that "in accordance with an option reserved to the lessee to cancel the lease on Jan. 1, 1915, the Great Western Mining Company on Oct. 26, 1911, formally notified the lessors that the company elected to cancel the lease." It is not without interest to observe the coincidence of dates involved in this information and that conveyed in the succeeding paragraph of the report, which paragraph begins: "On Oct. 26, 1911, the United States Government filed a petition in the Circuit Court" against the Corporation under the act of July 2, 1890. The statement is made that it is the intention to mine sufficient ore prior to Jan. 1, 1915, to balance the royalty payments exacted under the terms of the lease, which involve a progressive annual minimum. This will make a total of 26,474,320 tons of ore to be shipped under the lease prior to its relinquishment, of which 7,832,137 tons was shipped up to Dec. 31, 1911.

## Boston Meeting of the American Electrochemical Society.

The annual meeting of the American Electrochemical Society will be held in Boston, Mass., from April 18 to 20.

Meetings for the reading and discussion of papers will be held on Thursday morning, Friday morning and afternoon, and Saturday morning. One of the sessions will be held at Harvard University, the others at the Massachusetts Institute of Technology.

The sessions of Friday will be devoted to a symposium on electric conduction to which Messrs. W. C. Bray, C. J. Davisson, H. M. Goodwin, C. A. Kraus, S. C. Lind, S. J. Lloyd, D. McIntosh, H. N. McCoy, R. A. Millikan, T. G. Nutting, O. W. Richardson, and E. Weintraub will contribute papers. An illustrated lecture on conduction through gases will be delivered by Prof. Cross on Thursday evening before the smoker.

For the other professional sessions papers have been announced by Messrs. W. D. Bancroft, C. W. Bennett, R. Beutner, W. F. Bleeker, D. S. Cole, C. G. Fink, F. A. J. FitzGerald, C. Hambuechen, Carl Hering, A. R. Johnson and W. R. Woolrich, O. L. Kowalke, F. C. Mathers and O. R. Overman, W. Lash Miller, J. W. Richards, L. E. Saunders, W. Sykes, R. Turnbull, and J. R. Wilson.

There will be a smoker on Thursday night and a dinner on Friday night and on Thursday afternoon excursions, one to Worcester (Norton Company works and Steel Corporation electric-furnace plant), the other to the General Electric Works in Lynn. Hotel headquarters are at the Hotel Lenox.

## The Iron and Steel Market.

On the whole, it may be said that a measurable improvement occurred in the iron and steel trade during March, but the improvement was disappointing, as throughout the first two months of the year a very noteworthy improvement was expected upon the approach of spring. Such improvement as has occurred has only been in spots, there being no semblance of sustained buying. For instance, the agricultural implement trade is normally expected to be placing heavy specifications at this time, while the current condition is that its specifications are light. Apparently the implement trade is deterred more by the political outlook than it is encouraged by the prospects of excellent crops.

The price situation in steel products is altogether unsatisfactory, in that there is no response to demand. The finished steel industry is really experiencing a heavy demand, sufficient to engage 80 or 85 per cent of existing capacity, against an average of less than 70 per cent during 1911, yet prices show no resistance, and for several commodities are absolutely the lowest on record. This last statement is true of shafting, railroad spikes, merchant pipe, boiler tubes and sheets. These products are selling at fully as low a level as last November, and in some cases at lower levels. At that general low point a number of products dropped to the lowest level on record. Others, which remained a trifle above their minimum in the historic low period of 1897-8, scored advances in December, but have lost portions of these advances.

The backbone of demand at this time is the steel car trade. A heavy buying movement started late in October and enabled the car shops to enter the present year with orders for more than 75,000 all-steel or steel underframed cars on their books. The program then adopted was to operate at between 50 and 60 per cent of capacity, which would meet the double desideratum of producing at a low cost and at the same time hold the operating organizations together. The shops have a total capacity of about 1000 all-steel or steel underframed cars per day, and in the first three months of the year have been producing at an average of not more than 500 cars a day. Since January 1 they have booked new orders to the extent of

more than 20,000 cars, fully equal to their current output, and are now in the position of being able to operate at the present gait until almost the close of the year without any new business.

Pig iron is being produced at the rate of about 27,000,000 tons a year, against a capacity of about 34,000,000 tons, and 23,650,000 tons actually produced in 1911. The steel industry proper is operating at about 85 per cent of capacity.

The prospective suspension of mining in the anthracite and union bituminous coal districts March 31 has had the effect of creating a heavy demand for coal and advancing prices of both coal and coke. By the middle of the month Connellsburg furnace coke for spot shipment commanded \$2.10 at ovens, against \$1.50 in November and an average of about \$1.85 in January. A suspension of mining is certain, and is likely to affect both production and consumption of iron and steel, fully as much the latter as the former, so that the disturbance promises no good. Many by-product coke plants, which depend upon union-mined coal, will probably have their output materially reduced. The Connellsburg region is non-union, but cannot increase its output materially.

### Pig Iron.

Early in March a sharp burst of activity occurred in Bessemer pig iron in the central west, sales aggregating 120,000 tons in a few days, the United States Steel Corporation taking 30,000 tons, the Youngstown Sheet & Tube Company 80,000 tons, and the Republic Iron & Steel Company 10,000 tons. The general price was \$14. valley, which had been the market, but the figure was slightly shaded in some of the transactions, and for 55,000 tons of the iron taken by the Youngstown Sheet & Tube Company ore was given in partial payment. The market immediately stiffened to \$14.25, and at this figure half a dozen steel casting concerns bought from 1000 to 2500 tons each. Basic iron in the central west was also active. Several interests together absorbed the entire stock of warrant iron at Midland furnace, some 50,000 tons which had been an incubus on the market for a year, while furnaces also effected important sales, and the market advanced sharply, closing the month at \$13. valley, for prompt and \$13.25 to \$13.50 for extended delivery. The low point had been \$12.25, valley, in January. Southern iron steadily stiffened during the month, though without large transactions, and closed with \$10.25, Birmingham, the minimum on nearby deliveries, and \$10.50 demand for second quarter. In other markets the movement was limited, but the tendency is toward higher prices. Current quotations are: Foundry iron, \$10.25 to \$10.50, Birmingham; \$14.75 to \$15, Philadelphia; \$13.75, Buffalo; \$14, Chicago; \$13.25, valley; Bessemer, \$14.25, valley; basic, \$13 to \$13.50, valley, malleable, \$13, valley.

### Steel.

Inquiry has not been heavy for unfinished steel. Some sheet and tin mills have made desultory inquiries for second half sheet bars, without business resulting. As in other branches of the market, there is a wide range in the anxiety of the mills to sell, some being ready to cut prices while others do not quote at all. Minimum quotations are on the basis of about \$19.50 for billets and \$20 for sheet bars, but these figures are only approximate. Bessemer is scarce at Pittsburgh, but freely offered at Youngstown, while with open-hearth material the reverse is the case.

### Finished Material.

Current prices, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill except Colorado.

Plates, tank quality, 1.10 to 1.15; Chicago, 1.25 to 1.28 cents. Shapes, 1.12½ to 1.15 cents.

Steel bars, 1.10 cents; 1.25 cents, Chicago.

Iron bars, 1.25 cents, Pittsburgh; 1.22½ to 1.25 cents, Philadelphia; 1.15 to 1.20 cents, Chicago.

### The Western Metallurgical Field.

#### Remodeling the Smuggler-Union Mill.

Important alterations are under way at the milling plants of the Smuggler-Union Mining Company, at Telluride, Colo., which, when completed, will result in very materially improving the system of ore treatment. The mills owned by this company are shown in the accompanying illustration. The two stamp-amalgamation-concentration mills are shown at the right and left of the picture, with the cyanide plant in the center. The stamp mill at the left is partly in operation by lessees, and the other stamp mill and cyanide plant are operated by the company. It is in the latter two plants that the changes are being made.

The stamp mill is being remodeled principally as regards concentration. Formerly bumping tables were used; but these have been removed and replaced by Wilfley tables. The flow of ore in the remodeled mill will be about as follows: The stamps will crush through 16-mesh screen and deliver the pulp onto amalgamating plates. Following the plates are Pierce riffles, and from these the ore will flow to a hydraulic classifier. The products from the coarse spigots will be treated on Wilfley tables, while the classifier overflow will be thickened in Callow cones and concentrated on vanners. A feature of the treatment will be the use of Hardinge conical mills for regrinding table middling.

The tailings from the stamp mill will be sent to the cyanide plant for further treatment. The overflow of the Callow cones and the vanner tailing will be combined into one product, all of which is fine enough for agitation in cyanide solution. The tailing from the Wilfley tables will contain sand and slime and will have to be classified.

Formerly the only cyanide treatment consisted in leaching sand tailing. The stamp mill tailing was charged into tanks, the slime being allowed to overflow and pass to the creek. The sand charge was then leached with cyanide solution. The new system will retain the old sand leaching plant, but will include, in addition, a modern slime agitation-filtration plant. The Wilfley tailing will be classified into sand and slime by means of Akins classifiers. The sand will be treated as before, but the extraction should be materially improved and the time of treatment lessened, owing to the complete removal of the slime. The classified slime will be combined with the vanner slime, and collected in 40-ft. tanks fitted with Dorr continuous slime thickeners.

An interesting problem was presented in designing the slime plant. The pulp is sent to the cyanide plant in a very dilute condition, possibly 25 or 30 parts of water to 1 of dry slime. This necessitated getting rid of a large quantity of water as completely as possible, for in the subsequent continuous cyanide treatment it would be necessary to throw away as much barren cyanide solution as might enter the system with the slime pulp. Hence, thickening and filtration were necessary preliminary processes before cyanide solution could be added.

Four of the old sand leaching tanks have been converted into Dorr thickeners. Three of these will be used in dewater-

ing the original slime pulp, and the fourth as a continuous decanter of cyanide solution from the treated pulp. The thickened pulp will be sent to the dewatering filters, which are of the continuous revolving-drum type known as the Portland. As the dewatered slime is discharged from the filters it is mixed with cyanide solution and sent into the agitation system.

Four Pachuca tanks have been erected, adapted to either intermittent or continuous agitation. The tanks are 15 ft. diameter and 45 ft. high, with 60-deg. cone bottoms. After agitation, the pulp will be pumped to the fourth Dorr thickener, the overflow of which will go to the zinc boxes. The thickened pulp will be pumped to an equalizing tank which will



SMUGGLER-UNION MILL, TELLURIDE, COLO.

feed other Portland filters, where the final recovery of cyanide solution will be made.

Thus the process will consist in thickening and filtering original slime pulp, agitating in cyanide solution, followed by further thickening and filtration of treated pulp. The system presents an interesting experiment, for by first crushing and concentrating in water, and then filtering the tailing before cyanidation an excellent opportunity will be afforded to wash cyanides out of the pulp, and the idea may prove of value to other San Juan properties producing ore not directly amenable to cyanidation.

The present system of zinc box precipitation will be retained, and the "waste" box solution will be discarded in sufficient quantity to counterbalance the water introduced into the system with the dewatered slime pulp.

#### Company Reports.

*The Florence-Goldfield Mining Company*, about which there has been so much speculation since the loss of the mill by fire in December last, has issued a report covering the last calendar year. It appears that the revenue from milling operations in January, 1911, was less than for several months previous, owing to reduced grade of the ore, and that this general condition prevailed until August, when the value of the ore fell so low that it was a question whether it would be worth milling. This continued through October, and in November drastic measures were taken to raise the grade of ore by close sorting and by taking some ore from several dumps. In the

latter part of November the mine began to make a better showing, and it was believed that the improvement would be permanent.

On the evening of Dec. 2, however, a fire of unknown origin started in the boiler room of the mill and completely destroyed the plant. The value of the property destroyed, after taking care of depreciation, which always has been the company's policy, stood on the books Dec. 1, 1911, at \$295,980. The insurance on the same was \$83,500, leaving a net loss of \$212,480. The destruction of the mill has so crippled the company that it is deemed best to spend every effort on mine development before reconstructing the mill.

During the year 1911 there was treated a total of 48,847 dry tons of ore, of gross value of \$298,193, yielding bullion to the value of \$251,561, representing a recovery of 83.96 per cent. No concentrate was made, as a system of local treatment was devised which permitted extraction by amalgamation and cyanidation.

The *Giroux Consolidated Mines Company* has issued its annual report for the last calendar year, covering operations at the properties at Kimberly, Nev. In addition to detailed reports on mine development, the report contains announcement of the company's arrangement for ore treatment.

There is now fully developed in the Morris-Bunker Hill mine 4,010,000 tons of porphyry ore of an average assay value of 2.14 per cent copper. There is also a large quantity, probably several million tons, of partly developed ore of at least an average assay value of 1.73 per cent copper. In the whole, it is estimated that there is a total tonnage of fully and partly developed ore amounting to 10,291,000 tons assaying 2 per cent copper; 6,599,000 tons fully and partly developed ore assaying 1.65 per cent copper, and several million tons of lower grade ore carrying at least 1 per cent copper. It is estimated that the precious metal value of these ores will amount to not less than 1 cent per pound of refined copper.

The company has a concentrating plant of 500 tons capacity at Kimberly, but it has not been operated since 1909 on account of a shortage of water. In view of the ample water supply developed in the Alpha shaft, sufficient to supply a concentrating plant of several times the capacity of the present unit, it was thought it might be advisable to operate the concentrator instead of selling the ore. Investigation, however, showed that at least eight months' time would be required to build additions to plant in order to bring it up to 1000 or 1200 tons daily capacity, and in the meantime it was thought wise to bring the mine to the producing stage. Consequently the company entered into a contract with the Steptoe Valley Company (Nevada Con) for the treatment of from 900 to 1200 tons daily. The rates and charges are such that the cost of producing and marketing refined copper will be about 9.5 cents per pound. The contract commences May 1, 1912, and may be terminated by either party on one year's notice.

The *Temiskaming Mining Company* has issued its fifth annual report covering the calendar year 1911. Owing to a change in the formation encountered in the mine, it is believed by the general manager that the life of the mine has been materially shortened. As a consequence of this condition the company has purchased a controlling interest in the North Dome property at Porcupine and expects to continue as a profit-paying company. The Temiskaming mine at Cobalt still contains reserves, but it is difficult to estimate them with accuracy.

During 1911 the mine produced 24,783 tons of ore at a cost of \$6.85 per ton, being 46 cents less than for the previous year. Hand sorting played an important part in the production, 6559 tons being handled, producing 139 tons of shipping grade, 6058 tons of mill ore and 362 tons of waste. The total cost of this work, including bagging and shipping the high-grade ore, was 41 cents per ton, being 14 cents per ton less than last year. The assay value of the shipping grade ore was 4801.4 oz. silver per ton. The highest grade shipment contained 6414 oz.

silver per ton. The average price received for silver was 53.9 cents per ounce, as compared with 54.3 cents last year. No payment was made in 1911 for cobalt, nickel or arsenic; on the other hand, penalties were exacted for arsenic in some cases.

The mill treated 34,720 tons of ore containing 685,120 oz. silver, producing 770 tons of concentrate containing 546,743 oz. silver. This gives a concentration of about 45:1 and a recovery of 80 per cent. The stamp duty was 3.13 tons per 24 hours. The total milling cost was \$3 per ton, being 86 cents less than last year. This cost is divided as follows: Mill, \$1.44; crushing, 0.30; ore house, 0.08; aerial transportation, 0.07; laboratory, 0.09; smelting, 1.02. Total mine and mill cost, \$10.57 per ton.

*The Arizona Copper Company, Ltd.*, announces in its annual report under date of Feb. 23, 1912, that the proposal for the erection of new reduction works and power plant has been unanimously approved, and Consulting Engineer Ricketts has been instructed to proceed with detailed plans and other preliminary work. Early in January a contract for the supply and erection of steel work was accepted. Ground was broken on Jan. 15.

*Ray Consolidated Copper Company*.—In a report covering the fourth quarter of 1911 it is announced that the rate of production during the period was about 100,000 tons per month. It is estimated that at the end of March the production would reach 4500 tons per day. The average grade of ore for the quarter was slightly less than 1.75 per cent. It is expected that the grade will improve during the first quarter of 1912 so that it will average 2 per cent copper. The cost of mining has been about 76 cents per ton and of milling about 57 cents.

The fifth section of the concentrator was constructed with certain alterations and improvements resulting from the operation of the first four sections, and the first four have been similarly altered. No further sections will be put in operation until a capacity greater than 5000 tons per day is required. The rate of recovery has averaged about 70 per cent. Since the middle of February concentrates have been delivered to the new A. S. & R. Company smelter at Hayden.

*The Chino Copper Company* has issued its first quarterly report covering the fourth quarter of 1911. Since Jan. 1 two sections of the mill have been in operation, treating about 800 tons each per day. The ore is partly oxidized and the recovery is not as good as it will be later on—clean sulphide ore. The recovery has been at the rate of 30 lb. of copper per ton at a cost of 8.3 cents per pound. The completed plant of five sections should be in operation in July.

## CORRESPONDENCE.

### Pinch-Effect Furnace.

To the Editor of Metallurgical and Chemical Engineering:

Sir: Concerning the new type of electric furnace in which the force developed by the pinch effect is used for circulating the metal in the resistors (see this journal, vol. IX, page 277 and 371), the belief has been expressed by critics both in this country and abroad, that there would be a serious wear on the refractory walls of the holes in which the heat is generated, caused by the rapid flow of heated metal through them.

Will you therefore permit me to say in your journal, in justice to that furnace, that trials were recently made to determine the extent of this much talked-of wearing away.

In a furnace which had been run daily with bronze, the last charge was allowed to freeze and the furnace was then demolished when cold so as to expose the frozen cores in these holes. It was found impossible to detect any increase in size. The surface of the refractory material (a special form of magnesite) forming the walls of these holes, was found to be clean, even and smooth, as though it had been sand-papered.

A similar test was then made with a steel furnace; it had been running daily, not only on steels but also on some steel alloys, the heating of some of the charges having been very

intense. Again no measurable wear could be detected. On the contrary there seemed to be even a slight diminution of the cross section of the holes. This was apparently due to a hard, smooth, clean, black coating which forms on the surfaces of the holes, presumably from the oxides and other impurities in the iron combining with the magnesite.

Hence the fear about this much talked-of wear seems to have been premature and needless. But even if there should be some wear, as there may be in course of time, it can be taken care of to some extent by regulating the transformer to give a greater current at a lower voltage; such regulation should be provided in any case on account of the varying resistivities of the metals due to different compositions and different temperatures. Should it exceed the limits of the transformer it can be taken care of by lengthening the holes by tamping a layer of refractory material over the bottom, as is frequently done in repairing steel furnaces. If the ratio of length to section be maintained the same, the heat produced will remain the same. And as the pinch force would generally be more than ample, a slight reduction in this force due to an enlarged cross section will be of no importance.

Another unexpected result is that the expansion and contraction of the metal and the lining for short holes seems to be so nearly the same that no rupture of the core takes place in cooling down over night; the furnaces were started repeatedly in the mornings on the remnant of the frozen charge of the day before.

*Philadelphia, Pa.*

CARL HERING

#### The Crystallization of Carbon by Artificial Means.

*To the Editor of Metallurgical and Chemical Engineering:*

SIR:—The artificial production of diamond has doubtless been attempted many times, in a somewhat indifferent manner, but apparently the effort has never been followed up with any degree of persistence, and certainly not to a successful issue in a commercial sense within the knowledge of the public. Our available information on the subject is limited to the published accounts of certain experiments made by a few noted chemists some years ago; and what may have been done by others the result of whose experience has never been disclosed can only be conjectured.

The reports of hidden secrets and stories of wonderful discoveries in this direction which have now and then become current have heretofore invariably been discredited, but manufacturing chemists are now producing many substances of natural origin by artificial means; and a diamond even of the most perfect formation is composed solely of one ingredient, if we may use the term, and that constituent is a simple element which is otherwise of the commonest occurrence.

When these facts are considered in view of the many achievements of modern science, and the marvelous feats of applied chemistry which have revolutionized the industries of the world, it is strange, indeed, that the artificial production of a material now so excessively costly and which might otherwise be of great use in the mechanical arts has been so long delayed; and so far from being incredible, the news of its attainment would in reality be less wonderful than the apparent fact that it has not already been accomplished.

If it is feasible to undertake the construction by synthetic combination of a complex substance of vegetable origin, and of such peculiar nature and specific properties as india rubber, it is surely within the province of industrial chemistry to attempt the formation of a simple mineral by the transposition of an element from the amorphous to the crystalline form.

So far, however, as published accounts go, it would seem that the experiments hitherto tried for this purpose have been performed upon a very small scale, entirely inadequate for the production of decisive results, and by methods which are only partly capable of establishing the desired conditions. The means employed being some simple and expedient quickly devised and always abandoned without further attempt. And

yet it is an admitted fact that a number of these trials have produced traces of diamond material, very small particles, it is true, merely minute specks, but crystallized carbon in character, and this is diamond.

Moissan, in France in 1893, formed a saturated solution of carbon by heating melted iron to a temperature of 3000 deg. F. and adding all the carbon the molten metal would take, and then plunged the body of seething liquid into water; and Chroustchoff performed a similar experiment, using silver instead of iron. In both instances after the metals were dissolved away by an acid there remained at the bottom of the containing vessel a gritty black powder which proved to be graphite mixed with minute particles of crystallized carbon.

In these experiments advantage is taken of the fact that iron or silver at their boiling points, and also several other metals, notably tungsten and manganese when highly heated, will assimilate as much as 6 per cent of carbon, completely dissolving the element but precipitating it more or less upon a reduction of temperature; and it was Moissan's idea that the pressure produced by the contraction of sudden cooling would cause the disengaged carbon to crystallize.

Friedel, about the same time, heated to a temperature much below redness, in a closed vessel, a mixture of highly carbureted iron and sulphur, forming sulphide of iron, and as in the cases previously noted, the carbon was left in the form of a dust containing particles of extreme hardness and exhibiting the properties of diamond.

The experiment of Moissan is generally regarded as a demonstration of the theory that the diamonds of Nature were formed from melted and compressed carbon; that is, from carbon liberated and liquefied by some reaction at a very high temperature within the interior of the earth or some portion of its crust, and cooled under an intense pressure caused by the contraction of a mass of solidified mineral matter in which the body of fused carbon was embedded. But the analogy is very imperfect for the reason that carbon derived from its solution in molten iron is deposited at a temperature below its point of perfect fusion, and is for the most part precipitated (in consequence of this fact) in an amorphous condition. There evidently was, however, a partial fusion of some part of the disengaged carbon, and to this circumstance may be attributed the existence of the minute crystals found in the residuum.

In Friedel's experiment, however, there was no pressure and practically no heat, as the reaction was accomplished at a temperature but little above the melting point of sulphur, and we might be misled into the belief that the crystallization observed in this case was effected without the aid of heat or pressure, but the conclusion would be erroneous, as the crystals found were previously formed in the carbonized iron and the action of the sulphur was practically no more than that of a solvent, removing the iron by combination and leaving the carbon unchanged.

In estimating the practical value of these experiments for the purpose of further development it may be said that while they were conceived in recognition of a theory which is probably correct, they do not, as we have already seen, embody the conditions demanded by that theory, and can not therefore be made the basis of a successful process.

Professor Hanney as long ago as 1880 also experimented upon this problem by heating a hydrocarbon liquid in contact with magnesium in a closed iron tube, but the tubes usually burst, and after a number of vexatious trials the effort was abandoned. The trouble was caused by the fact that the heat applied to the outside of the tubes weakened the metal to the point of rupture long before the required temperature was even approached. It was Hanney's conclusion that the diamond could only be formed at an extremely high temperature and under an almost incalculable degree of pressure. Such conditions as might be imagined to exist within the impenetrable walls of the earth's crust at a depth of 2 miles below the surface. But it is highly probable that if a body of pure car-

bon fused and thoroughly liquified by a sufficiently high temperature could be maintained without evaporation complete crystallization would ensue as the element cooled.

It is impossible, however, to secure such conditions in the open air by any means whatever, as the melting point of carbon is away above its temperature of vaporization, and were the requisite degree of heat applied in this way complete evaporation would take place without fusion, and just what condition would be assumed by the condensed vapor is somewhat uncertain, but we may fairly conclude that it would be some form of the inevitable black powder and not diamond.

We know, however, that evaporation is retarded by pressure and that the boiling point of a liquid is partly dependent upon the pressure to which it is submitted; and it is, therefore, evident that if we could melt a body of carbon under a sufficient pressure to raise the boiling point above the temperature of fusion we would have secured the conditions which, there is good reason to believe, would result in the formation of diamond.

The melting point of carbon is said to have been estimated by Sir William Crookes at 2400 deg. C., and its boiling point at 3500; and that eminent authority is further quoted as saying, in effect, that at the melting temperature a pressure of 2551 lb. per square inch would be likely to promote crystallization. But these points can be definitely determined only by further experiment.

In regard to the available means of producing a temperature so high as the case requires, the electric arc at its highest power is incapable of melting carbon and is, therefore, out of the question, as is also the well-known thermit reaction as usually employed in an unconfined space, but Sir Andrew Noble is believed to have produced a temperature of 52000 deg. C. and a pressure of 50 tons per square inch by the explosion of picric acid in a closed steel tube, and here at least is a possible solution of the problem.

But if this statement is true it is highly probable that there are also many other explosive substances which are capable when ignited in a confined space of producing an equally high degree of heat. Possibly if the thermit reaction were conducted in this way, that is, burned in a closed steel tube under the pressure caused by the expansion of its own products, or of some added material, the temperature would reach a point equal to or even beyond that produced by the experiment of Noble. The pressure generated in this way, however, would be in no wise equal to that of the picric acid explosion, but it might reach 5 tons per square inch, and this should prevent the volatilization of a body of carbon embedded within the ignited mass and fused by the heat of the reaction.

In the accompanying drawing a mechanism proposed for these experiments is shown in the sectional view, in which *a* is a cylinder of vanadium steel 26 in. in length with a bore of 4 in. and having an outside diameter of 8 in. Taper threaded steel plugs are fitted to the ends. The side walls of the cylinder as well as the ends are lined with graphite or some form of carbon, 1 in. thick in the sides and 2 in. thick at the ends. This leaves a circular cavity 2 in. in diameter and 12 in. long for the reception of the charge. The detonator may be a fulminate cap, the insulated terminals of which pass out through a very small hole in the center of one of the end plugs, as shown in the figure, in which *b* is the carbon lining, *c* the charge, and *d* the detonator.

The object of the lining is twofold. It forms an insulation against the immediate escape of the heat and thereby promotes a higher temperature and a more prolonged effect, and also protects the walls of the cylinder and the ends of the plugs, and as a further precaution against the weakening effect of the heat upon the material of the cylinder the device when used is to be exploded under water. The charge may consist of picric acid, or of thermit powder mixed with, say, 10 per cent of graphite or lampblack. In the experiment of Noble there was evidently a production of free carbon resulting from the dis-

sociation of the elements of the explosive; but there was no more than a trace of crystallization in that case by reason partly of the small size of the apparatus used. The heat being dissipated so quickly that there was not sufficient time for the carbon fused in an atomized condition to collect in an integral body. But if these materials should fail in this case also, there is a very large number of other carbon-bearing substances which are capable of disassociation under similar conditions and which might yield the desired result. The following is a list of a few combinations that are known to be of this nature in a greater or less degree, and from which a successful operation might be secured:

Carbide of calcium is decomposed at a red heat in the presence of nitrogen, the latter uniting with the calcium and releasing the carbon.

Hydrocarbon liquids are decomposed by chlorine, a portion of which combines with the hydrogen and another portion with part of the carbon, leaving the remainder of the carbon free.

When carbon bisulphide is heated in contact with copper the metal appropriates the sulphur and throws down the disengaged carbon. Doubtless some other metals would have a similar effect, and conditions might be devised where-in a reaction of this kind would become so energetic as to assume the form of a violent explosion, evolving heat and generating sufficient pressure to crystallize the deposited carbon.

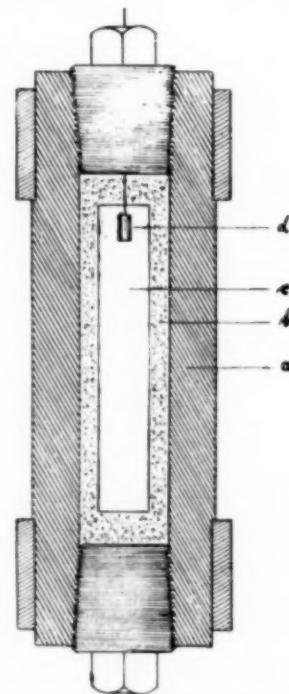
Phosphorus is also said to be capable of decomposing the bisulphide, and it would be easy to arrange means for conducting this reaction safely under a very high pressure and a considerable degree of heat.

Acetylene is a hydrocompound surcharged with carbon and easily decomposed, but when submitted to compression is of such a violent nature that its manipulation is extremely hazardous, and no experiment should ever be undertaken with it unless the circumstances are known to be such that no harm can come to the operator in the event of a premature explosion. If, however, the cylinder shown in the drawing could be filled with this gas previously liquified by cold, and then plugged up perfectly tight and placed in a body of water, the imprisoned liquid would explode spontaneously when its temperature reached the normal condition of the atmosphere or of the water in which the containing cylinder was immersed, and the nascent carbon thus disassociated would be placed under such extreme conditions of heat and pressure that crystallization might fairly be expected to result; that is, if the assumed theory be correct, and the process of Nature in the formation of diamond can possibly be duplicated by any means within the reach of art.

We might go on much further in this way enumerating combinations and chemical reactions possibly capable under the conditions proposed of resulting in the formation of diamond, but it is hoped that enough has been said to at least suggest a successful experiment to those who have the means and mechanical facilities, and that such a trial may subsequently lead to the development of a process for the production of crystallized carbon upon a commercial scale.

Oroville, Cal.

S. M. HOWELL.



SUGGESTED ARRANGEMENT.

### The American Chemical Industries.

To the Tariff Board we are indebted for a very interesting volume on some of the chemical industries of this country. It is entitled "**Chemicals, Oils, and Paints. Glossary on Schedule A.**" An analysis by paragraphs of Schedule A of the Tariff Act of August 5, 1909, and a statistical survey of the industries affected thereby, with a comparison of tariff laws since 1883. Prepared under the direction of the Tariff Board, transmitted to the Ways and Means Committee of the House of Representatives, February 7, 1912" (274 pages).

Being a congressional document submitted under the authority of the Tariff Board, the name of the author is not stated in the volume. However, at this place there is no reason to omit it, but every reason to congratulate Dr. BERNARD HERSTEIN for having produced a very interesting volume.

"Up to the year 1883 chemicals were distributed among the various tariff subdivisions of the respective tariff acts, although

in 1842, then changed in the same year to 5 cents per pound, with several changes subsequently. Paints in oil and also soap shared substantially the same fate; crude saltpeter, on the other hand, was kept almost without exception on the free list and the same is true of brimstone. Sulphuric acid is first mentioned in the law of 1824, carrying the duty of 3 cents per pound; reduced in 1824 to 1 cent, put on an ad valorem basis in 1846; on the free list in 1861, and, after intervals, again in 1872 and 1894. Muriatic acid appears for the first time in the act of 1832, boracic acid in 1841, most of the other common acids in 1846, alum in 1876, aluminum sulphate in 1862, aluminum itself 10 years later, in 1872. Barilla (natural soda) appears first in 1816, carbonate of soda in 1832, and soda ash in 1841. One year later (in 1842) bleaching powder is specifically mentioned, while anilin colors are made dutiable at 25 per cent ad valorem in the law of 1862, five years after their introduction into industry. These few instances may suffice.

"The United States census of 1880 had established the fact

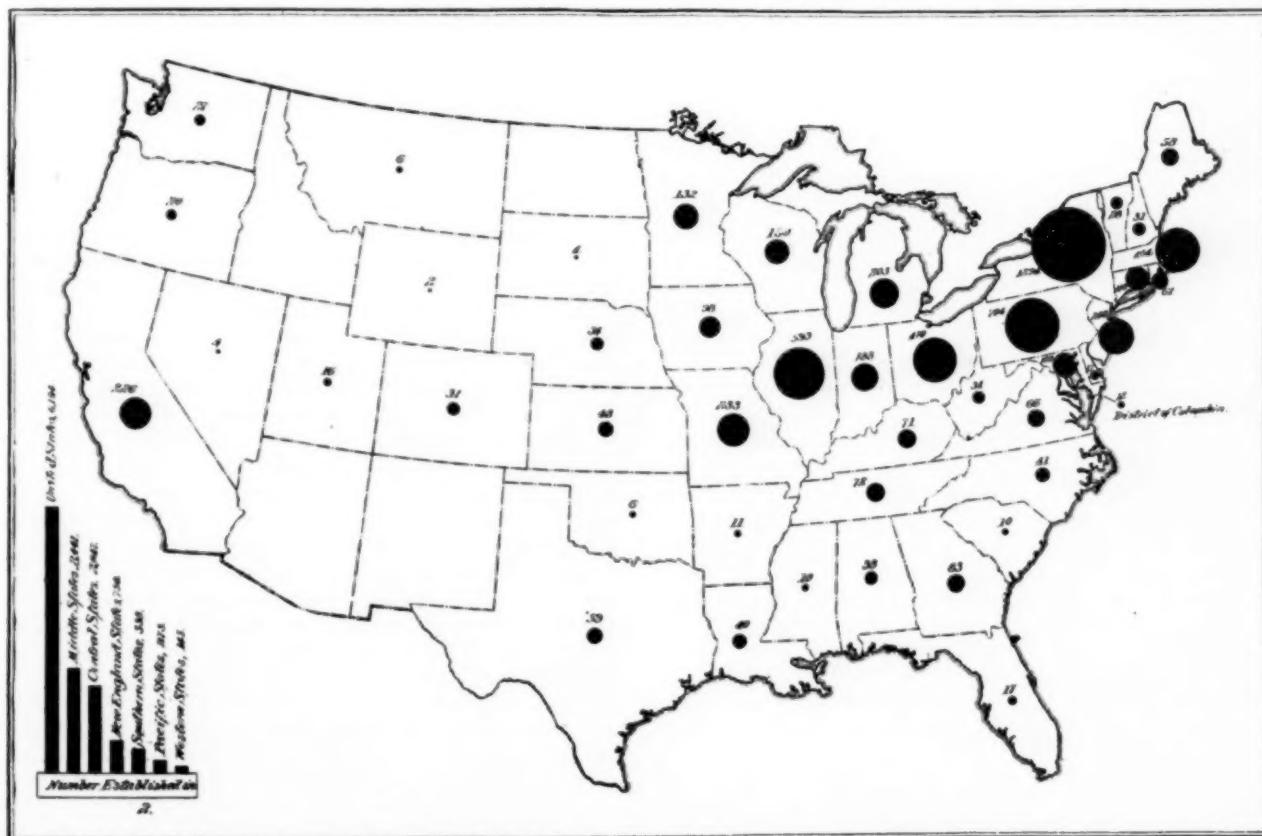


FIG. 1.—GEOGRAPHICAL DISTRIBUTION OF CHEMICAL ESTABLISHMENTS AFFECTED DIRECTLY BY SCHEDULE A.

classified principally under 'Sundries.' The tariff policy pursued toward chemicals varied, of course, with the vicissitudes of changing political opinion and practically every one of the seventy-odd tariff acts enacted in the United States since 1789 affected in some way one or another commodity properly regarded as chemical. With their increasing commercial or industrial importance more attention was bestowed upon these articles, and the list of special items increases continually as time passes. A comparison of the different tariff acts with reference to the time of the first introduction of, and the subsequent changes in, the rates of duty imposed upon the various chemicals would furnish a great deal of interesting historical material as to the development of the chemical industry especially in the United States.

"The very first act of July 4, 1789, imposes upon indigo a duty of 16 cents per pound, which was gradually raised to 50 cents in 1812; after several changes it was put on the free list

that chemical manufacturers occupied a prominent position among the industries of the country, and after the Tariff Commission of 1883 had recommended the subdivision of the proposed tariff law into generic schedules, each to comprise as much as possible a specific industrial group, chemicals, oils, etc., owing to their importance and their basic relation to other industries generally were assigned a group to themselves at the head of all others. Schedule A has since then remained the schedule devoted to chemicals, oils, paints, etc., and the general scheme of classification devised originally has in the main been followed in all subsequent revisions. Specific rates were changed or were altered to an ad valorem basis or inversely: single items were eliminated or added, put on the free list or made dutiable again in accordance with the policies of the tariff framers. . . .

"On account of the great and continuously increasing number of chemical products now on the market, their diversified

character and other distinctive features, a reclassification of chemicals on lines more indicative of their industrial and fiscal importance is made periodically by other countries, and it seems that in the United States much uncertainty and vexatious litigation could be eliminated if for the same reasons and to meet the same purposes a general reclassification were instituted at proper intervals."

The first part of the book (147 pages) is a Glossary on Schedule A, discussing the chemicals enumerated in the 83 paragraphs of this schedule, and giving in each case a concise elementary statement of the method of manufacture, with statistical data on home production and imports.

Schedule A comprises the following materials: acids, alcoholic compounds, alkalies, chemical compounds and salts, alumina, ammonia compounds, argols, blacking, bleaching powder, blue vitriol, charcoal, bone and blood char, borax, camphor, chalk, chloroform, coal-tar products, cobalt oxide, collodion and pyroxylin compounds, coloring for brandy, etc., coppers, drugs, barks, etc., ethers, extracts, blues and gelatinous, glycerin, indigo, ink and ink powders, iodine, iodoform, licorice extract, chicle, magnesia, oils, opium, coca leaves, barytes, ferrocyanide blues, blanc-fixe, blacks, chrome colors, mineral pigments, orange mineral and red lead, ultramarine blue, paints, pigments and varnishes, zinc chloride and sulphate, Paris green and London purple, lead compounds, phosphorus, bichromates, caustic potash, potassium iodide, saltpeter, prussiates of potash, medical preparations, corrosive sublimate and calomel, druggists' preparations, plasters, perfumery, santonin, soaps, soda compounds, moss and sea grass, sponges, strychnine, sulphur, sumac, vanilia.

Schedule A comprises, therefore, quite a heterogeneous mass of compounds, and on the other hand covers only part of the chemical industry and none of the metals.

The geophysical distribution of the chemical establishments affected directly by Schedule A is given in an interesting manner in Fig. 1, giving both the number of establishments by states (1) and by grand divisions (2). New York with 1292 establishments is far in the lead, followed by Pennsylvania, Illinois, Ohio, Massachusetts, New Jersey, etc., the list being closed by the purely agricultural states like Wyoming, South Dakota, Nevada, Oklahoma, Montana, with less than 10 establishments each, while Arizona, Idaho and North Dakota do not possess any establishments engaged in the manufacture of chemicals affected by the present tariff law.

The second section of the book (100 pages) gives historical sketches of the evolution and present status of the chemical industries in the United States, Germany, England, France and Canada.

"The beginning of chemical manufacturers in the United States, even those included only within the narrow definition commonly assumed, dates back to a very early period of our history. The first sulphuric acid plant using lead chambers was erected in Philadelphia in 1793 by John Harrison, who only a few years later introduced the method of refining the acid from a platinum still and claimed for himself the credit of having been the first to introduce this very important innovation. The manufacture of white lead by Wetherill had even preceded this undertaking, and this, too, was located in the Quaker City, which was early an important center of industrial and scientific activity. Other chemical establishments followed, particularly those manufacturing drugs and medical compositions, pigments and extracts for tanners and dyers.

"From Philadelphia the industry spread to Baltimore and gradually to New York and the New England states, but it was confined exclusively to the making of certain chemical specialties with no industrial connection or unity between them. This is not surprising in view of the purely agricultural status of the United States in those days.

"But even after manufactures began to be established here conditions changed but imperceptibly, the number of new chemical establishments increasing only very slowly as they were

able only with difficulty to withstand the pressure of the English competition, which at that time was world supreme and had every advantage in cheap supply of materials and superior knowledge and experience in the use of manufacturing processes. This apparently was the reason why the manufacture of soda ash by the Le Blanc process never gained a foothold on this side of the Atlantic and why this most powerful factor for the creation and expansion of a chemical industry in the United States was missing.

"In the second half of the nineteenth century progress was quickened. Petroleum refining, the increased demand for fertilizers, the development of the textile, tanning, paint, soap and other industries required a prompt and large supply of, and insured a satisfactory market for, specific chemical products, which American enterprise attempted by degrees to satisfy. The census of 1860, like all other census up to 1880, paid no particular attention to chemical manufactures, but by counting only such items as clearly belong within the group of manufacture of chemicals proper we find enumerated in its statistics 99 establishments which may be so classed, with an output valued at \$5,251,225 and employing 1636 wage-earners who received in wages the sum of \$561,672.

"These figures do not include potashes or lyes, medical compositions and pigments." A better appreciation of the status of the chemical industries of those days can be got from special tables prepared by the author from the original data as furnished by the census of 1860. These show that in 1860 there existed 3998 establishments for the manufacture of chemicals and allied products, with a total value of \$91,696,156 of the products. "The total number of manufacturing establishments counted in 1860 was 140,433, and the total value of their product amounted to \$1,885,861,676, so that the chemical industries were represented with 2.99 per cent in number of establishments and 4.89 per cent in value of product respectively."

"The two decades following from 1860 to 1880 were decisive periods in the history of the United States from both a national and economic point of view. The rearrangement which took place in the entire structure, economic and political, stimulated wonderfully the growth of manufactures, which during these 20 years almost doubled in the number of its establishments and trebled in the value of its products, the record for 1880 being 253,852 establishments and products valued at \$5,369,579,191, mining industries not being counted. During the time the population of the United States increased from 31,443,321 to 50,155,783, or about 60 per cent.

"The chemical industries for various reasons did not share proportionately in this development of manufactures generally. Yet they had grown considerably in importance, their economic character showing a marked change, as evidenced in the value of output per establishment, which increased from approximately \$23,000 in 1860 to \$87,000 in 1880. The importance which the chemical industries had then already attained in the economic life of the Union is also manifested by the fact that the census authorities for 1880, for the first time in the history of census taking, saw the necessity of treating the chemical manufacturers as a single unit in a special report."

The number of establishments for the manufacture of chemicals in 1880 was 1349, with \$85,400,000 invested capital and \$117,000,000 total value of products.

The later development of the chemical industries of this country is shown clearly in Table I, giving quantities and values of products made by establishments engaged in the manufacture of chemical products from the census reports for 1890, 1904 and 1909.

An examination of this table illustrates sufficiently the status of the chemical industry in the United States according to the latest available data.

From a discussion of the import and export trade it is concluded "that the manufacture of so-called staple products, especially inorganic, is carried on on a scale fully commensurate with the demand. The same may be said of certain special

TABLE I.—CHEMICALS PROPER, DETAILED SUMMARY—PRODUCTS.  
(As per census of 1909, 1904, and 1899, from proof sheets of the census of 1909.)

	Quantity	Value.					
		1909	1904	1899	1909	1904	1899
Total Value		\$117,688,887	\$75,222,249	\$46,746,752			
Acids, value	pounds						
Acetic	do	51,963,788	27,001,322	24,945,558	11,926,389	7,583,059	3,161,726
Boric	do	5,554,414	6,956,896	2,684,935	1,136,134	537,542	396,323
Citric	do	2,102,206	2,265,631	(*)	295,739	727,190	198,212
Hydrofluoric	do	4,790,963	2,932,358	698,000	777,200	598,718	(*)
Muriatic	do	128,394,736	127,502,682	116,675,109	214,657	151,218	34,890
Oleic	do	13,337,717	(*)	(*)	1,171,082	1,180,910	1,015,915
Phosphoric	do	25,702,606	991,050	(*)	680,015	68,541	(*)
Other, value					505,791	68,541	(*)
Sodas, value					7,145,771	4,518,940	1,516,398
Soda ash	tons	646,007	518,789	386,361	21,412,855	16,858,929	11,596,915
Sal soda	do	76,441	56,870	63,231	10,361,756	8,202,292	4,768,383
Bicarbonate of soda	do	82,800	68,867	68,185	972,585	792,248	779,166
Caustic soda	do	112,152	80,159	78,779	1,151,031	1,135,610	1,324,843
Borax	do	20,154	20,882	5,637	4,230,954	2,924,182	2,917,955
Other, value					1,766,910	2,122,808	502,480
Potashes					2,565,619	1,681,789	1,304,088
Alums, value					88,940	563,489	174,476
Alum, cake		1,866,570	5,113,706	3,764,806	2,578,842	2,126,612	2,013,607
Potash alum		26,884,880	(*)	(*)	273,711	(*)	(*)
All other, value		7,939,702	(*)	(*)	128,623	(*)	(*)
Coal-tar products, value					2,176,508	(*)	(*)
Coal-tar distillery products, value					2,673,327	844,817	1,322,094
Chemicals made from coal-tar distillery products, value					2,462,330	340,641	809,830
Cyanids, value					212,997	504,176	512,264
Yellow prussiate of potash		3,510,208	5,027,264	6,140,406	1,941,893	1,179,104	1,584,923
All other, value		5,999	(*)	(*)	463,983	683,277	993,514
Bleaching materials, value					1,477,910	495,827	591,409
Hydrogen, peroxide		9,403,717	(*)	(*)	1,635,046	777,750	492,086
Bisulphites	tons	5,999	(*)	(*)	850,417	(*)	(*)
All other, value					202,504	(*)	(*)
Chemical substances produced by the aid of electricity, value					582,125	777,750	492,086
Calcium carbide	pounds	121,946,967	(*)	(*)	17,920,409	5,896,632	1,305,368
Caustic soda	tons	18,158	(*)	(*)	2,984,001	(*)	(*)
Chlorates	pounds	11,568,915	(*)	(*)	984,779	(*)	(*)
Hypochlorites	tons	68,016	(*)	(*)	940,525	(*)	(*)
All other, value					1,506,831	(*)	(*)
Plastics, value					11,540,273	5,896,632	(*)
Pyroxylon plastics	pounds	6,206,177	(*)	(*)	7,180,172	4,755,761	2,099,400
All other	do	10,234,928	(*)	(*)	5,389,819	2,857,093	1,970,387
Compressed or liquefied gases, value					1,790,353	1,898,668	129,013
Anhydrous ammonia		11,802,076	(*)	(*)	4,969,805	2,787,689	1,215,011
Carbon dioxide		47,238,267	35,991,627	(*)	2,503,315	1,173,184	448,157
Laughing gas		72,675	(*)	(*)	2,317,808	1,343,966	696,164
Oxygen	gallons	4,777,977	(*)	(*)	33,689	(*)	(*)
All other	pounds	364,014	(*)	(*)	98,150	(*)	(*)
Fine chemicals, value					16,843	270,539	70,690
Alkaloids	ounces	3,482,493	4,949,525	3,387,522	10,940,346	9,145,853	4,220,339
Gold salts	do	42,544	59,969	8,594	3,188,691	2,925,789	1,743,264
Silver salts	do	2,027,719	1,743,882	1,252,604	430,944	449,864	90,145
Platinum salts	do	1,561	19,068	7,312	726,222	683,761	499,345
Chloroform	pounds	1,861,435	616,670	396,540	19,123	175,682	54,600
Ether	do	1,177,886	660,783	263,238	472,759	165,604	98,070
Acetone	do	6,723,886	1,300,395	1,638,715	199,448	334,935	129,876
All other, value					703,575	161,320	178,666
Chemicals not elsewhere specified, value					5,199,584	4,248,898	1,420,373
Glycerin	pounds	33,986,974	18,791,997	15,383,798	29,888,839	16,959,484	17,560,790
Epsom salts	do	47,785,318	15,935,837	6,072,309	4,838,826	2,345,205	2,012,886
Blue vitriol	do	810,958	50,100	7,500,000	357,728	145,801	45,966
Copperas	do	24,199,526	8,815,059	14,097,905	37,626	2,500	375,000
Phosphate of soda	do	116,126,671	12,018,815	3,478,350	71,081	28,061	58,581
Tin salts	do	12,992,233	9,573,719	4,677,471	1,194,546	904,679	470,159
Zinc salts	do	43,204,652	(*)	(*)	1,477,486	(*)	(*)
Other Chemicals: Value					21,411,520	13,289,416	14,493,644
By-products and residues sold to other industries: Value					4,530,024	5,743,070	(*)

<sup>1</sup> In addition, products to the value of \$8,329,440 were produced by establishments engaged primarily in the manufacture of products other than those covered by the industry designation, including: Acetic acid, 4,959,985 pounds, value \$200,740; hydrofluoric acid, 2,051,951 pounds, value \$79,722; muriatic acid, 74,805,743 pounds, value \$587,253; oleic acid, 2,939,346 pounds, value \$165,091; stearic acid, 5,094,774 pounds, value \$399,386; other acids valued at \$49,530 (not including acids reported by manufacturers of explosives and fertilizers); sal soda, 10,822 tons, value \$184,297; other sodas, 75,902 tons, value \$1,835,278 (including sodas reported by manufacturers of paints and varnishes and fertilizers); potashes, 14,293,552 pounds, value \$525,054; alums, 49,450,260 pounds, value \$443,513; coal-tar distillery products valued at \$1,610,792; hydrogen peroxide, 521,851 pounds, value \$20,124; bisulphite bleaching materials, 3,062,000 pounds, value \$23,650; other bleaching materials valued at \$20,703; pyroxylon plastic, 292,582 pounds, value \$282,560; anhydrous ammonia, 167,770 pounds, value \$40,923; carbon dioxide, 454,354 pounds, value \$19,262; laughing gas, 24,500 pounds, value \$4,900; oxygen, 23,826,325 gallons, value \$79,319; other compressed or liquefied gases, valued at \$9,072; chloroform, 8,250 pounds, value \$4,779; acetone, 2,007,560 pounds, value \$210,287; glycerin, 1,022,920 pounds, value \$123,472 (not including 52,518,919 pounds, value \$6,864,603, reported by manufacturers of soap); blue vitriol, 20,362,647 pounds, value \$720,636; copperas, 3,031,566 pounds, value \$53,372; phosphates of soda, 310,588 pounds, value \$27,034; and zinc salts, 4,312,988 pounds, value \$103,503.

<sup>2</sup> In addition, products to the value of \$3,063,397 were produced by establishments engaged primarily in the manufacture of products other than those covered by the industry designation, including: Muriatic acid, 47,018,080 pounds, value \$431,938; stearic acid, 1,750,000 pounds, value \$140,000; hydrofluoric acid, 1,217,518 pounds, value \$71,668; other acids, valued at \$146,716; sal soda, 1,763 tons, value \$79,561; caustic acid, 14 tons, value \$668; other sodas 14,200 tons, value \$363,765; alums, 33,074,349 pounds, value \$532,185; coal-tar distillery products, value \$238,645; bisulphite bleaching materials, 536 tons, value \$11,937; glycerin, 520,000 pounds, value \$52,000; ether, 193,628 pounds, value \$92,466; Epsom salts, 1,350,000 pounds, value \$13,500; blue vitriol, 107,160 pounds, value \$5,994; copperas, 81,816 pounds, value \$586; and tin salts, 1,103,222 pounds, value \$188,301.

<sup>3</sup> Not reported separately.

<sup>4</sup> Not reported.

manufactures, as for instance, fertilizers, soaps, paints, standard medicinal preparations, and some few others, the development of which has reached a state permitting of a considerable export trade. On the other hand quite a number of chemical manufacturers offer very much room for improvement in quantity, variety and in some instances also in quality of products made. Among these, and in the first place, must be mentioned the production of coal-tar dyes and coal-tar products generally, the manufacture of so-called 'fine chemicals,' high-priced ex-

tracts, and essential oils, and a great number of other products, consumed in relatively large quantities for a variety of purposes. Nevertheless, taken as a whole, the manufacture of chemicals proper during the intervening 10-year period has shown marked progress."

Details are given of the import and export trade with diagrams showing the fluctuations from year to year.

Quite extensive chapters are then devoted to the chemical industries of Germany, England, France and Canada.

### The Electrolytic Recovery of Tin from Tin Sheet Scraps in Italy.

Our contemporary, *Rassegna Minearia Metallurgica e Chimica*, gives an interesting review of the Italian detinning industry with reference to official statistical reports of the Industrial and Commercial Public Survey of Italy.\* According to this report there are three large Italian plants operating the electrolytic process. After several commercially unsuccessful attempts and the failure of one large establishment in the Valle d'Aosta, the following three plants were built in recent years and are now operating with complete success. The Dossmann Company in Pegli, the Eugenio Guec in Arenzano, and the Societa Electrochimica Vesuviana in S. Giovanni a Tettuccio.

The first-named factory has been operating for five years, the first few years being devoted to such experimental work as every new industry requires. These investigations are partly still being continued, especially for the object of decreasing the cost of the process and for utilizing the by-products, like tin hydrate, which the company tries to work up into tetrachloride. We may give here the description of the process in an abbreviated form.

The electrolyte is a sodium hydrate solution of well determined concentration and temperature, the anodes being tin sheet scrap and the cathodes being iron plates. The tin is dissolved in the form of sodium stannate and deposited at the cathode as metal, while the soda is regenerated. The electrolytically produced oxygen dissolves more tin from the anode, while the hydrogen liberated at the cathode causes the tin to be precipitated in spongy form. The best results are obtained with only 2 or 3 volts per cell and a very low current density, which is never over 0.75 amp per square decimeter (1.16 milliampere per square inch). Experience shows that the most favorable concentration of the bath is from 10 to 12 per cent total alkalinity, the free alkali not exceeding 7 per cent. The carbonate must not exceed 3 per cent and the stannate not 5 per cent. The electrolyte is held at 60 deg. to 70 deg. C. and kept in continuous circulation. This is absolutely necessary in order to bring fresh tin into the immediate vicinity of the cathodes and to keep the electrolyte homogeneous.

The plant in Pegli contains nine tanks arranged in three parallel series of three tanks each. Each tank contains six anodes and six cathodes. The anodes consist of rectangular cages or baskets, made up of perforated iron sheets and contain from 100 to 130 lb. of tin sheet scrap. They measure about 30 in. x 35 in. x 18 in. The cathodes are iron plates 39 in. x 35 in. The bath is made up of a solution of 7 per cent to 10 per cent caustic soda, which is kept in circulation by a pump and reheated during this circulation in special heating tanks.

In order to be completely detinned the tin sheet scrap has to stay three hours in the bath, which means in view of the extra time for charging and discharging that each anode basket can be used for six operations in 24 hours. The electrolyte keeps at sufficiently uniform composition for two days and has then to be regenerated by addition of a certain amount of fresh solution. As an approximate rule, it can be assumed that about one-tenth of the solution has to be replaced every week.

After complete detinning, the contents of the anode baskets are combined in packets of 150 lb., and for thorough removal of the last traces of tin they are passed through a bath of the same composition as the electrolyte at 100 deg. C. and are finally washed twice in cold water. The better the tin is removed the better is the iron scrap for steel making.

The spongy tin is removed every 12 hours by carefully lifting the cathode and dipping it in water, the water tanks being held on a truck. Here the tin is held under water all the time. This precaution is necessary to prevent oxidizing of the tin by air. After the spongy material has been completely freed of soda by careful washings it is compressed in a small hydraulic press into 6-lb. cylinders, which afterward

are melted in a furnace with sealed tubes. The sponge contains about 50 per cent metallic tin and 50 per cent ashes. These latter are mixed with powdered carbon and reduced in a little open-hearth furnace with another output of 70 per cent metal.

It is necessary to regenerate the tin contained in the discarded part of the solution. This is done by saturating the bath with carbonic acid under pressure, by which process the tin hydrate is precipitated. The sodium carbonate solution is then boiled in order to remove the excess of carbonic acid and finally treated with lime in order to regenerate the caustic soda. The slimes produced by introducing carbonic acid contain 15 per cent of tin. They, in turn, give an output of 40 per cent of metal in the reducing furnace.

The electric power is furnished by two direct-current dynamos, one producing 700 amp, the other 1100 amp, both at 27 to 30 volt. They are driven by a 35 and 50-hp power gas engine, respectively. The current is used mainly for electrolysis and for driving a 25-hp motor for the centrifugal pumps, the press, etc.

Every week this plant treats 50 tons of tin sheet scrap, or 2500 tons per year, with an average force of 20 to 26 laborers and one superintendent. It is equipped, however, for the treatment of double the quantity. The technical results are quite satisfactory. About 75 per cent of the tin is recovered in the metallic state. To this has to be added 6 per cent from the metal contained in the slimes of 15 per cent contents, and in the ashes of 30 per cent contents; considering, however, that for the tin regenerated from the slimes and ashes only half the price is paid, it appears that about 80 per cent of the tin can be sold. It is to be considered, however, that the tin contains up to 2 per cent of lead and a little iron and is, therefore, sold at a price 5 per cent below the price of tin produced from minerals.

The plant of S. Giovanni a Tettuccio, established in 1909, was built for treating 2000 tons per year. The same process as described above is applied here because it is found to be more successful than the use of an acid electrolyte and because the detinning with dry chlorine under pressure is considered to have not yet proven a commercial success. (The Italian official who compiled this report seems to have been unaware of the fact that in other countries the dry chlorine detinning process is rapidly advancing and has proven superior to the electrolytic process.)

The plant contains three batteries of electrolytic baths; each battery comprises three tanks, connected one to the other, with a capacity of 3 cu. m each, corresponding to approximately 85 cu. ft. The electrolyte is circulated by a pump which, from the level of each battery, lifts the liquid up into a huge vessel where it is reheated by steam coils. From this vessel the liquid is run back through the batteries. Each process lasts three hours and the operation is continuous day and night.

The electric energy is derived from a 27-volt, 1200-amp dynamo; the current density is little more than half that used in the first-named factory.

The tin obtained in spongy deposits is compressed in cylinders of 3 to 4 cu. decimeters (190 to 250 cu. in.), is then melted in tube furnaces and gives 70 per cent metal and 30 per cent ashes. From these ashes 70 per cent to 75 per cent metal is obtained by reduction in a small open-hearth. The rest remains in the slags, which are sold for further treatment in England. The residues are washed after complete detinning, compressed under 300 atmospheres into square packets of small volume and sold to steel makers. They still contain, however, 1/10 per cent of tin and a certain quantity of lead from the solder of the tin sheet scrap, this quantity of lead varying with the nature of the raw material.

The electrolyte is regenerated twice a week in the same way as mentioned above.

The operating details in the third factory are said to be the same as in the two others.

R. A.

## Miscellaneous Cyanide Experiments.

BY LOUIS COHEN.

In accumulating data pertaining to the testing of ores for cyanide treatment, I have noted some experiments, the results and comparison of which have afforded me a great deal of interest. They may prove to be of similar interest to others engaged in cyanide operations. A few of these experiments I will briefly describe, as they present some unusual as well as unexpected results. They demonstrate the effect of slight differences in the mechanical application of the methods of operation, and also the effect of the addition of other chemicals in connection with potassium cyanide. The increase in extraction of precious metals, the saving in cyanide, the gain in time of treatment which may be an economy in mill construction, and a reduced cost of treatment are some of the factors which enter into these results.

These experiments were made with the idea in mind of their practical application; that is, the working costs must be such as to make the use of the process profitable, and the mode of operation simple. The experiments also show the great variation in the results on the same ore under slightly different conditions, as well as the marked similarity in the results obtained from ores with entirely different characteristics.

It may be noted in some of the experiments that the extractions are so low and the cyanide solution of such strength as to seem unpractical. Further, the consumption of cyanide may seem exceedingly high. In all these cases it must be remembered that the experiments have been selected from a series, in order to offer comparisons along certain lines, and not to be considered apart from the series, except for purposes of comparison. For example, in one case it may be thought that the use of a preliminary wash would have saved an excessive consumption of cyanide, and such is absolutely true.

The experiments made to determine the most economical cyanide strength are not given. The object of this comparison is only to bring out the difference in the amount of cyanide consumed under varying conditions, and not to show whether it was practical to treat a particular ore with a certain strength of solution with its resulting high cyanide consumption.

### Effect of Aeration of Slime.

The object of aeration in the cyanidation of ores is, of course, to supply oxygen which is essential for the formation of a soluble salt of gold and potassium cyanide.

When the gold has had sufficient oxygen present to form a soluble salt and still does not dissolve, it is considered not amenable to cyanidation. In other words, the mechanical condition of the gold may be such that it will not readily dissolve in cyanidation solution; or, a chemical combination may exist that is not acted upon by the cyanide solution. Then again, there may be a combination of these two; for example, a thin film on the surface of the gold may prevent the cyanide solution from attacking it.

The gold that remains in the ore and is still in the insoluble form after the completion of cyanidation is lost. The soluble gold is recovered to a considerable degree by washing and displacing the gold-bearing solution with barren solution or water. The insoluble form in the tailing may be amenable to some other form of recovery, as amalgamation or concentration; or it may be stored in a tailing pile for future treatment by cyanidation.

In the following two experiments, two kinds of gold are distinguished. The gold soluble in potassium cyanide which is held in the moisture of the wet slime, is termed dissolved gold; and the gold in the slime which is not acted on by the cyanide solution is termed insoluble gold.

It is taken for granted that aeration has been quite complete by agitation with air in a Pachuca tank, the ratio of solution to dry slime being 2:1 or greater. The aeration results on thick pulps will be shown to be better than those of the thinner

pulps usually treated in Pachuca tanks. The experiments were conducted on quite different ores, and show remarkable results.

### EXPERIMENT NO. 1. LEACHING TEST.

The ore was one generally known as an oxidized ore containing a small amount of heavy mineral in the form of pyrite. It assayed very low in gold. The sand and slime had been separated by classification, as is customary in plants which do not make an all-sludge product.

The sand was subjected to leaching in tanks, and the slime was kept in suspension preparatory to treatment. After agitating and aerating continuously for a number of hours, the slime pulp was filtered, leaving about 38 per cent moisture in the cake. Air was then drawn through for one minute, followed by a cyanide solution.

The slime previous to the aeration of one minute contained insoluble gold to the value of \$1.22 per ton. After the aeration of one minute, followed by the cyanide solution, the slime contained insoluble gold to the value of \$0.76 per ton. The difference of \$0.46 in gold per ton was changed from insoluble gold to dissolved gold, due to the aeration and the application of the second cyanide solution.

### EXPERIMENT NO. 2. LEACHING TEST.

The ore was a sulphide assaying high in gold. The insoluble gold in the slime assayed \$0.44 per ton after cyanide extraction had been partly completed. The solution had been in contact with the slime for a number of hours, and aerated by agitation in a Pachuca tank. After filtration, air was drawn through the slime for one minute, followed by a cyanide solution. The amount of gold remaining in the insoluble form was nil. The difference in extraction of \$0.44 gold per ton was due to the aeration and application of the second cyanide solution.

The results just given, confirm each other on two quite different ores. There exists a favorable condition for aeration in slime, containing a solution of cyanide in the form of 30 per cent moisture. Conditions for aeration are not so favorable in thin slime pulp with a ratio of two parts solution to one of dry slime. The factor of a second cyanide solution with a lower gold content enters into consideration, as well. The fact, however, is quite pronounced that after a number of hours' agitation and aeration, the effect of one minute more aeration materially affected the extraction.

In order to find out whether pure gold in potassium cyanide solution would be similarly affected, the following experiment was tried. It substantiates in a measure the importance of the time element in comparing the effect of air in solution with that of air on the surface of the liquid.

### Effect of Oxygen at the Surface and Beneath the Surface of a Solution.

A solution of 8.2 lb. potassium cyanide per ton was made alkaline with sodium hydroxide equivalent to the extent of 1.5 lb. protective alkalinity. To some of this solution in a flask was added gold leaf in small particles, allowing some to float on the surface and the balance to sink. The flask was agitated gently to facilitate solution of the gold, but not sufficiently to cause the floating gold to sink. The gold on the surface of the solution was dissolved completely in 5 minutes.

The flask was then agitated more violently, but the time required to completely dissolve the gold which had sunk beneath the surface of the solution was 240 minutes. It took, therefore, 48 times as long to dissolve the submerged gold as it did that portion which was in contact with the air on the surface.

### Effect of Aeration on a Sand Charge.

#### EXPERIMENT NO. 1. LEACHING TEST.

A heavy pyritic ore assaying \$18 in gold was crushed to pass a 40-mesh screen. A portion was subject to continuous leaching with cyanide solution. Another charge was leached intermittently with cyanide solution, air being drawn through the ore between alternate leaching periods. A 10-lb. potassium

cyanide solution was used in this test after the same preliminary washes in each case. The cyanide solution, in the ratio of 1:1 of ore, was allowed to percolate through the charge six times in each case. The leaching time was 72 hours in the continuous experiment, and 18 hours in the intermittent; these being the actual periods of contact with the cyanide solution.

The results were as follows: The tailing from the continuous treatment assayed \$11.20 in gold per ton, corresponding to an extraction of 37.8 per cent; the cyanide consumption was 13.5 lb. per ton of ore. The tailing from the intermittent treatment assayed \$8.40 in gold per ton, corresponding to an extraction of 53.3 per cent; the cyanide consumption was 7 lb. per ton of ore.

This experiment was performed on coarse ore, and at the time of aeration there was present as small a quantity of solution as possible. It was the intention to find out whether gold in coarse ore would act in a similar manner to gold in slime.

The results probably would show a greater difference than they do if the results were expressed in terms of insoluble gold as in the slime experiments, but for certain reasons they are expressed as total gold in the tailings. In this comparison three points are prominent, all of which are in favor of the aerated charge, viz., approximately one-half greater extraction, one-half less cyanide consumption, and one-fourth the time of treatment.

#### EXPERIMENT NO. 2. LEACHING TEST.

An oxidized ore containing no sulphide minerals was crushed to pass an 8-mesh screen. It assayed \$8.80 in gold per ton.

One portion was treated by continuous leaching, while another charge was leached intermittently, with alternate periods of leaching and aeration as described before. A 3-lb. cyanide solution was used, after the same preliminary washes in each case, at the ratio of 1:1 of ore. This solution percolated through the charge four times in each case. The leaching time in the continuous test was 43 hours, and 6 hours in the intermittent; these being the actual times of contact with the solution.

The results were as follows: The tailing from the continuous experiment assayed \$7.60 in gold per ton, corresponding to an extraction of 13.6 per cent; the cyanide consumption was 3.2 lb. per ton of ore. The aerated tailing from the intermittent experiment assayed \$7 in gold per ton, corresponding to an extraction of 20.5 per cent; the cyanide consumption was 1.6 lb. per ton of ore, and lime consumption amounted to 7.6 lb. per ton of ore.

The result of the experiment was that the aerated charge gave approximately one-half better extraction in one-seventh the time, with approximately one-half the consumption of potassium cyanide and lime as compared with continuous leaching. In these two experiments the ores were as different in character as it was possible to select, and yet the results show striking similarity in extraction and cyanide consumption. I do not mean to imply that these conditions would obtain with all ores, as aeration sometimes is not as essential for good extraction as it appeared to be in these cases.

#### Effect of Picric Acid on Cyanide Extraction.

##### EXPERIMENT NO. I. AGITATION TEST.

A heavy sulphide ore containing a large percentage of pyrite was crushed to pass an 80-mesh screen. The ore assayed \$8 in gold per ton. It was agitated for 19 hours in a solution containing 3 lb. potassium cyanide per ton, with lime as protective alkali. Another charge was treated with the same solution, but with the addition of 0.8 lb. picric acid per ton of ore.

The tailing from the regular cyanide treatment assayed \$5.20 in gold per ton, corresponding to an extraction of 35 per cent of the gold value with a cyanide consumption of 6 lbs. per ton. The tailing from the picric acid treatment assayed \$3 per ton, corresponding to an extraction of 62.5 per cent of the gold value, with a cyanide consumption of 5.6 lb. per ton.

The result of this comparison shows an extraction approximately double, and a cyanide consumption approximately 10 per cent less in the case of the picric acid addition as compared with the regular cyanide treatment.

#### EXPERIMENT NO. 2. AGITATION TEST.

An ore assaying \$16.20 in gold per ton was crushed to pass an 80-mesh screen. It was agitated in a solution containing 1.8 lb. potassium cyanide per ton, with lime as protective alkali. Another charge was agitated with the same solution, but with the addition of 0.5 lb. picric acid per ton of ore. The time of treatment and conditions were the same in each case.

The tailing from the regular cyanide treatment assayed \$2.40 in gold per ton, corresponding to an extraction of 85.2 per cent; the cyanide consumption was 0.4 lb. per ton, and lime consumption 4.6 lb. per ton. The tailing from the picric acid treatment assayed \$2 per ton, corresponding to an extraction of 88 per cent of the gold value, with a cyanide consumption of 0.4 lb. per ton and a lime consumption of 4.2 lb. per ton.

With this particular ore it is noted that the difference between the two extractions (about 3 per cent) is not as great as in the preceding test. Other tests with this particular ore showed that neither aeration nor oxidizers were essential to its treatment.

#### EXPERIMENT NO. 3. AGITATION TEST.

An ore assaying \$8.80 in gold per ton was treated with a 10.5-lb. cyanide solution, with no protective alkali. Another charge was treated with a similar solution, with the addition of 0.5 lb. picric acid per ton of ore. The time of treatment and other conditions were the same. The tailing from the regular cyanide treatment assayed \$2 in gold per ton, corresponding to an extraction of 77.3 per cent. The consumption of cyanide was 1.2 lb. per ton. The tailing from the picric acid treatment assayed \$3.20 per ton, corresponding to an extraction of 63.6 per cent of the gold. The cyanide consumption was 1 lb. per ton.

The result of this experiment shows that the extraction in the straight cyanide treatment without a protective alkali was greater than that obtained when picric acid was used. The consumption of cyanide was practically the same in each case.

#### Effect of Lead Acetate on Cyanide Extraction.

##### EXPERIMENT NO. I. LEACHING TEST.

A heavy sulphide ore assaying \$18 in gold per ton was crushed to pass a 20-mesh screen. It was leached for seven days with a 2-lb. cyanide solution containing a protective alkali. Another charge was treated with a similar solution with the addition of 1 lb. lead acetate per ton of ore. The extraction by the simple cyanide treatment was 29.5 per cent of the gold, the cyanide consumption being 3.4 lb. per ton. The extraction by the treatment with lead acetate was 35.7 per cent of the gold, the cyanide consumption being 3.6 lb. per ton.

The result shows one-fourth greater extraction by the addition of lead acetate as compared with the simple cyanide solution.

#### Effect of Cyanide Solution on a Raw and Roasted Ore; Conditions of Roasting; Loss on a Copper Ore.

##### EXPERIMENT NO. I.

The raw ore, which was a heavy sulphide, was crushed to pass a 10-mesh screen. It assayed \$5.20 in gold per ton and 1.38 per cent copper. It was treated with a solution containing 5 lb. potassium cyanide per ton, 10 lb. of lime being added as protective alkali. The tailing assayed \$5.20 in gold per ton, showing no extraction. The cyanide consumption was 7.6 lb. per ton, and the corresponding lime consumption amounted to 5.4 lb. per ton.

##### EXPERIMENT NO. 2.

The ore was charged into a hot furnace and roasted at a low heat, and continued at a low, red heat until most of the sulphur was burned off. The final heat was raised until the

roast was sweet. The roasted ore assayed 0.03 per cent sulphur and \$6.80 in gold per ton. It was agitated with a 5-lb. cyanide solution, with the addition of 10 lb. of lime per ton. The tailing assayed \$1.40 in gold per ton, corresponding to an extraction of 79.4 per cent, the cyanide consumption being 5.2 lb. per ton and the lime consumption 3.4 lb. per ton.

## EXPERIMENT NO. 3.

The second charge of ore was put into a cold furnace, and the temperature gradually raised until the sulphur ignited. The furnace was kept at a low temperature until the sulphur ceased burning, and the final heat was then increased until the roast was sweet. The roasted ore assayed 0.02 per cent total sulphur and \$7.20 in gold per ton. It was then agitated with a 5-lb. cyanide solution, with the addition of 20 lb. of lime per ton. The tailing assayed \$0.60 in gold per ton, corresponding to an extraction of 91.7 per cent of the gold. The lime consumption was 11.4 lb. per ton.

For the purpose of better comparison the results of these three tests are tabulated below.

Exp. No.	Material	Sulphur Extraction Consumption lb. per-cent per cent Cyanide Lime		
		0.0	7.6	5.4
1	Raw ore .....	0.0	7.6	5.4
2	Ore roasted at high temperature .....	0.03	79.4	5.2
3	Ore roasted at low temperature .....	0.02	91.7	3.2
				11.4

The difference in extraction may be accounted for by the assumption that the gold was free and existed in the fissures between the crystals of mineral or in the fracture planes, and that it was not amenable to cyanidation owing to a possible surface coating. The condition after roasting was favorable to cyanidation, due to the removal of the protective coating. In experiment No. 3 this protective coating appears to have been almost completely removed. In experiment No. 2 the gold may have become partly re-coated. Whether these assumptions are correct, the fact remains that the results are quite remarkable and of considerable importance.

This ore contained 1.38 per cent copper, probably in the sulphide form. It is interesting to note that the consumption of cyanide in pounds per ton was 7.6, 5.2 and 3.2 respectively. This would seem to indicate that there existed in the raw ore a small amount of copper which readily consumed cyanide; that a part of this copper was changed in experiment No. 2 so that it did not attack the cyanide; and that in experiment No. 3 still more of the copper present as a cyanide in the raw ore was changed to another form, which seems to be more acid than either of the first two, as indicated by the increasing lime consumption.

## Effect of a Solution Without Protective Alkali Compared to a Solution Containing Protective Alkali.

An ore assaying \$8.80 in gold per ton was treated with preliminary water and lime solution washes, and leached with a 9.7-lb. cyanide solution for forty-eight hours. The solution, in the ratio of 2:1 of ore, was passed through the charge four times. Another charge was leached with a 10-lb. cyanide solution which contained no protective alkali. The tailing resulting from leaching with cyanide solution containing protective alkali, assayed \$5.20 in gold per ton, corresponding to an extraction of 40.9 per cent. Cyanide was consumed to the extent of 6 lb. per ton, and lime 19.3 lb. per ton. The tailing from the treatment without protective alkali assayed \$2.20 in gold per ton, corresponding to an extraction of 75 per cent of the gold. The consumption of cyanide was only 3.4 lb. per ton.

This was a remarkable test, for the extraction in the acid solution was almost double that in the alkaline solution. Further, the acid solution consumed approximately only one-half the cyanide consumed by the alkaline solution, and there was a saving of all the lime used in the first test. While this is extremely interesting, it probably would be dangerous practice

and should be attempted only with proper facilities, as great danger might attend the formation of hydrocyanic acid gas in the acid solution.

Denver, Col.

## Mechanism of the Formation of Troostite.

BY DENISON K. BULLENS.

At the Fifth Congress of the International Association for Testing Materials, held at Copenhagen in 1909, a list of definitions of the micrographic constituents of steel was adopted. The final report advocated the use of the term "osmondite" to cover the intermediary state in the conversion of martensite towards the more stable pearlite. That is, the generally recognized constituents troostite and sorbite are to be considered as intermediaries between martensite and osmondite on one hand, and between osmondite and pearlite on the other. That these two transition constituents can be readily recognized under the microscope, however, is clear to every student of metallography, and much objection has been heard concerning the "official" elimination of them.

The great importance of troostite is emphasized by the recent advances made in the scientific hardening and tempering of steels. These processes are no longer carried on in a haphazard fashion by many companies, but in conjunction with a careful study of the steel in question. This scientific investigation has naturally led to much research as to the nature of troostite; that is, the mechanism of its formation and its transition into the more stable pearlite.

Troostite has probably called forth a greater variety of opinions and theories from investigators than any other of the micrographic constituents of steel. A brief résumé is necessary to a thorough understanding of the subject at hand, and for this purpose the following have been taken from numerous English, German, and French periodicals.

Breuil<sup>1</sup> based his theory upon the proposition that troostite should be characterized by finely divided graphite. This has been entirely disproved by experiments involving specific volumes.

Boynton<sup>2</sup> has since admitted the error of his assumption that troostite is beta (pure) iron.

Rogers<sup>3</sup> considered it to be a solution of gamma iron, i. e., a mixture of either ferrite or cementite with a solid solution of the two in gamma iron. This theory has met with no favor.

Kourbatoff<sup>4</sup> considers troostite to be a solution of elementary carbon in alpha iron. This theory was also advanced by Sauveur,<sup>5</sup> who puts forward the following series of suggestions.

Austenite, a solid solution of carbon in gamma iron, on reaching the upper critical zone, has its iron transformed into the allotropic beta iron, forming martensite. In cooling still fur-



FIG. 1.—STRUCTURE OF ORIGINAL EUTECTOID STEEL X 50.

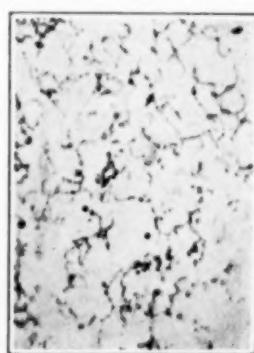


FIG. 2.—FIRST APPEARANCE OF TROOSTITE X 50

<sup>1</sup>Breuil, Sci. Papers, vol. ii., p. 623.

<sup>2</sup>H. C. Boynton, Journ. Inst. Iron & Steel, 1904, I., p. 262.

<sup>3</sup>F. Rogers, *ibid*, 1905, I., p. 484.

<sup>4</sup>W. A. Kourbatoff, Rev. de Metallurgie, 1905, II., 169.

<sup>5</sup>A. Sauveur, Journ. Inst. Iron & Steel, 1906, IV., 109.

ther, beta iron falls out of the beta iron solution (in the case of the hypoeutectoid steels). In passing through the middle critical zone, all the iron (being in the beta form) is changed into allotropic alpha iron, the alpha solid solution being troostite. And through the lower critical solution the troostite (alpha iron solid solution with carbon) is broken into its components, forming pearlite. Sauveur, however, does not insist upon a solution of elementary carbon, but allows the interpretation with the carbide in solid solution. Kourbatoff differed from Sauveur only in that he considered the carbon to be in elementary form, while the latter said "that it never has been shown conclusively that it was the carbon rather than a carbide which was dissolved in the iron."

Benedicks<sup>6</sup> concludes from his

researches that "troostite is a pearlite with ultra-microscopic small particles of cementite (containing also more or less hardening carbon). This is in keeping with the ease with which the troostite is affected by reagents, with its varying hardness—which lies between that of martensite and pearlite—and with the fact that troostite from a steel rich in carbon yields cementite when slightly heated, as Kourbatoff has shown. In all probability troostite is a transition *in situ* of martensite, which implies that the carbon it contains must be the same as in the martensite from which it originated. Troostite being more stable than martensite, the only possibility is that it is constituted of beta or alpha iron."

In a later paper<sup>7</sup> he advances the following additional arguments: a, the formation of troostite is not dependent upon the quenching temperature, but on the rate of cooling; b, the formation of troostite is accompanied by recalescence; c, the electrical resistance and specific volume are both the same as pearlite.

Fletcher<sup>8</sup>, in discussing Benedick's first paper, states that troostite might be classed as carbide-saturated beta iron, containing also about 0.9 per cent carbon. He was of the opinion that troostite was not so hard as martensite: a, because beta iron was softer than gamma iron; b, the carbide or cementite was in a finer state of division than in martensite. He suggests that sorbite is composed of two perlites,  $\text{Fe}_3\text{C}$  carbide in beta iron (troostite), and  $\text{Fe}_3\text{C}$  carbide in alpha iron (pearlite).

Campbell<sup>9</sup>, in an article on "The Constitution of Carbon Steels," explains the change from martensite to pearlite in the following manner: "The gradual change of the allotropic alpha iron is indicated (in the martensite) by a gradual loss of the needle-like martensitic structure and is replaced by the troostitic structure. Troostite is either a supersaturated solid solution of carbide of iron in alpha iron, part of which at least is under stress, or a mixture of solid solution of iron in allotropic iron with alpha iron and precipitated carbide of iron

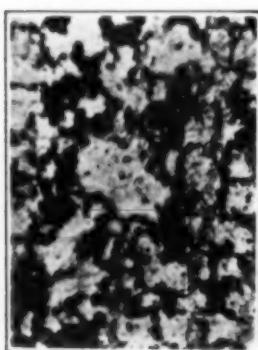


FIG. 3.—FURTHER INCREASE OF TROOSTITE X 50.

in particles too fine to be discernible under the microscope. With a rise in temperature the molecular freedom becomes greater and the carbide either precipitates or segregates into microscopic particles, and the internal stress of the alpha iron is relieved.

"We now have sorbite."

Kroll, in an article<sup>10</sup> remarkable for the tremendous amount of reasoning involved, takes up the subject of the iron-carbon system from an entirely different standpoint—that of crystallography—from that usually adopted. His nomenclature is worthy of notice, and part of which is given here. He divides cementite into two classes. First, "cementite," free, hexagonally crystallized carbide  $\text{Fe}_3\text{C}$ , insoluble in iron. Second, "regular cementite," to which he gives the new name of "solvite."

This latter carbide is the one present in solid and liquid solutions; dissolves gamma iron and is soluble in it; in a pure state, attached to and imbedded in austenitic grains, and at the same time often recrystallized in elongated regular cubes; quenched, it appears at most as pseudomorphs, consisting of hexagonal cementite. Troostite he defines as deposited solvite, more or less saturated with gamma iron. Regarding this latter he states that grounds were discovered for the assumption that the regular solution carbide (solvite) strives af-

FIG. 5 — MARTENSITIC STRUCTURE X 675.

ter saturation with gamma iron, thus forming the constituent troostite (the x constituent).

The general tendency of thought at the present time seems to be towards Benedick's proposition of an ultra-microscopic pearlite, regarding troostite as an aggregate. If this is true, there must be some point at which the ultra-microscopic particles become microscopic. There are, however, a large number of investigators who maintain that troostite is a solution of carbon (or carbide) in allotropic iron.

In either case, a rigorous microscopic investigation into the subject should throw additional light upon it. This will involve the use of as nearly pure steels as possible, and a heat treatment (the Metcalf test) which will give a continuous structure between and through the critical range. In other words, the study of the transition of martensite into troostite into sorbite.

The preliminary requirement is pure iron-carbon alloys, in which the influence of such elements as silicon, manganese, phosphorus, sulphur, etc., is reduced to a minimum. For this purpose, special steels were obtained from the Halcomb Steel Company, of Syracuse, N. Y. The two grades of eutectoid and hyper-eutectoid steels were investigated by submitting 6-in. lengths of  $\frac{1}{2}$ -in. rounds to the Metcalf test. That is, by heating one end to about 1050 deg. C., and quenching the entire piece in water. The pieces were then fractured lengthwise, polished, etched, and photographed. A solution of nitric acid in absolute alcohol was used in some instances, especially where it was desired to bring out the difference between martensite and austenite, as in Fig. 5. In the main, however, it was found that Kourbatoff's reagent gave the best results for the distinguishing of the troostite.

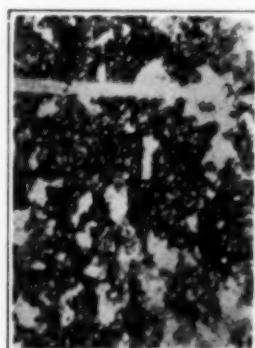


FIG. 4—FURTHER INCREASE OF TROOSTITE AND CHANGES INTO SORBITE AND OSMONDITE X 50.



FIG. 6—PRECIPITATION OF PRE-TROOSTITE X 675.

<sup>6</sup>C. Benedicks, *Journ. Iron & Steel Inst.*, 1905, II., 352.

<sup>7</sup>Ibid., 1908, IV., 220.

<sup>8</sup>J. E. Fletcher, *Journ. Iron & Steel*, 1905, II., 366.

<sup>9</sup>E. D. Campbell, *ibid.*, 1908, II., 321.

<sup>10</sup>A. Kroll, *Journ. Iron & Steel*, 1910, I., 382.

The composition of the steels used is given in the following table:

	ANALYSES.	Hyper-eutectoid.	Eutectoid.
Carbon	.....	0.92	1.48
Manganese	.....	0.14	0.16
Sulphur	.....	0.011	0.006
Phosphorus	.....	0.009	0.009
Silicon	.....	0.14	0.14

It will be seen from the above that the minor components are so low that they will probably have no effect upon the micrographic constituents.

A continuous photomicrograph of several feet in length was obtained from each piece, representing the transition from austenite to the original steel. In all cases these showed, under low-power lenses, the familiar structure known to all metallographists.

Fig. 1\* illustrates the structure of the original eutectoid steel, showing both sorbitic and pearlitic constituents.

The troostite—in hardened steel—first appears at the boundaries of the martensite crystals as a thin black line, as shown in Fig. 2, increasing to greater width and with the formation of globules, as in Fig. 3. It soon occupies the whole of the section, but also begins to change into sorbite or osmondite, as shown in Fig. 4.

This photomicrograph also shows the difference in hardness between the martensite yet remaining unchanged and the troostite, the latter being readily scratched with a needle. The same hardness results were also obtained with sections having a much smaller amount of troostite. Both the eutectoid and hyper-eutectoid steels in the earlier parts of the austenite transformation, when taken at 100 magnifications appear to have the same structure. But when we study the structure at very high magnifications—1350 diameters†—much more important features are observed, and throw much light on the mechanics of the transformation.

Fig. 5, so etched with nitric acid solution and photographed as to bring out the martensitic structure, gives a wonderful insight into the first appearance of the troostite. The development of austenite (white) into needles of martensite (gray) is as many other experimenters have noted. But furthermore, there may be seen darker splotches forming at the intersections of the austenite-martensite grains. These spots are surrounded by little dots which seem to merge and agglomerate into the spots, which are troostite. We have here the change or precipitation, of the martensite needles into the embryo "germs" of what we may term "pre-troostite." The section corresponds to one even less further advanced in the transformation than that shown at low magnification in Fig. 2 on page 205.

\*Cuts 1 to 4 are reduced in size one-half from the original photographs, which were 100 diameters magnification.

†The cuts 5 to 9 are reduced in size one-half from the original photographs, which were 1350 diameters magnification.

Figs. 6, 7, 8, and 9, etched with Kourbatoff's reagent, are photomicrographs of the hyper-eutectoid steel at 1350 diameters, taken at sections showing a larger growth of troostite, together with its resolution. The precipitation of the "pre-troostite" is again shown as dots upon the white background in Fig. 6, together with the larger globular masses of troostite. Fig. 7 is the same as Fig. 6, except that the print was so developed as to bring out the structure of the troostite areas. The troostite has the pebbly appearance noted by many investigators, the more fully resolved than usual, due to the exceptional magnification. We have here also the first appearance of free cementite, which appears to be thrown out as veins.

FIG. 9—REARRANGEMENT OF TROOSTITIC MASSES INTO "OSMONDITE" WITH FIRST APPEARANCE OF PEARLITE X 675.

Fig. 8 shows the pre-troostite in the martensite, areas of troostite, and larger amounts of free cementite. Here we have the free cementite with the familiar spine-like growth branching out from the main veins. It will be noticed that the free cementite appears only in the troostite.

Fig. 9, at a much further advanced stage in the transformation, shows the rearrangement of the troostitic masses into that which would be called "osmondite" by many, together with the first appearance of pearlite (lower left-hand corner).

We may now sum up the various transitions as follows:

- (a) Austenite.
- (b) Martensite, appearing as needles, characteristic.
- (c) Precipitation (or change) of infinitesimal particles of pre-troostite (elementary troostite) from the martensite.
- (d) Segregation of pre-troostite to the boundaries of the austenite-martensite grains, forming splotches (veins, under low magnification) and globular masses. Metallographically known as troostite.
- (e) Differentiation, almost immediately following the segregation, of the troostite into osmondite, together with the precipitation of free cementite in the case of hyper-eutectoid steels. The osmondite, from the pebbly appearance, is probably made up of the individual constituents ferrite and cementite.
- (f) Mechanical arrangement of osmondite into sorbite and pearlite.

From a study of these and other steels, as well as a close investigation of the photomicrographs of other experimenters, the author believes that the masses generally known as troostite may generally be termed osmondite; that the real troostite is that shown by the dots in the photomicrographs. The change after segregation is so rapid that the resolution of the segregated troostite takes place almost immediately, giving osmondite and cementite—in hyper-eutectoid steels.

Reading, Pa.

### The Important Factors in Blast-Roasting.

By H. B. PULSIFER.

(Concluded from page 159.)

#### 6. Greenawalt Down-Draft Pans.

A Greenawalt pan was successfully operated at the Modern plant, near Denver, Colo., in 1910. An account of the operating is given by Austin, Mineral Industry, 1910, p. 186, but the following may well be added:

Construction began in December, 1909, but various delays prevented operation until February, 1910. One pan only was in service almost continuously from then until the plant closed down in June, 1910.

The material sintered was copper concentrates and flue dust. This mixture was just such a one as others have not been able

to sinter successfully on account of the fineness of the material. Many of the first trials were flat failures, and good results came only when it was found that the moisture content was the vital point. Charges which could be ignited only in spots and utterly incompetent of complete sintering gave splendid sinters when properly moistened and thoroughly mixed. Depending on the fineness of the charge up to 15 per cent water had to be added.

The regular charge for sintering consisted of about 3 parts of flue dust with 1 of concentrates. As the flue dust contained only some 7 per cent. of sulphur the concentrates supplied most of the fuel value. Larger proportions of concentrates could be used, even up to 2 parts of concentrates, with 1 part of flue dust. All intermediate proportions made good sinters, but for the requirements of the regular work the first was used. The sulphur content varied from 12 to 15 per cent., and seldom exceeded 16 per cent.

At the Modern plant it was desired to sinter the flue dust and concentrates, not primarily to desulphurize. Accordingly no special attention was given to the sulphur elimination. The charges were dumped red hot and smoking, and as soon as the material had agglomerated. This eliminated about 50 per cent. of the sulphur, whereas longer treatment would eliminate 75 per cent.

With the comparatively low sulphur of the charge, its high silica content, and practical absence of lead, no ill effects resulted from forcing the draft. It was found that the time of operating could be materially reduced by raising the suction. From 1 hour with 12 inches of water suction the time fell to 45 minutes, with 16 inches, and finally to about 35 minutes with 20 inches suction. The Garden City fan, first used, proved unequal to the task; it had been designed to run at 3,000 r.p.m., and as run at 2,600 gave 16 inches suction, when speeded up to 2,800 it went to pieces. A rotary blower, next used, supplied the 20-in. suction, which would agglomerate a cake in about 35 minutes.

If lead to the extent of 2 or 3 per cent. happened to be present in the charge the sinter only improved in quality from the good fluxing properties of the lead.

To remove the fume and keep the blower in good condition a small stream of water was admitted into the blower. About 1,800 cu. ft. of air were required each minute.

Considerable fines, up to 15 per cent., were made when the cake from the pan was dumped onto the breaking grizzly below. This might certainly be improved upon for larger installations.

The process, as a whole, proved so successful and acceptable that a second pan had already been cast and a new and stronger wheel for the original fan was already on hand when the plant closed down. The second pan had a capacity of 2 tons per charge, as did the first.

For the blast furnace smelting the sinter proved of exceptional smelting qualities; its quick and easy fusion, its physical condition and fluxing properties rendered it available for irregularities in the furnace, where it came in demand to use instead of slag.

The second installation of a Greenawalt down-draft pan was made at the Midvale plant of the United States Smelting, Refining Mining Company early in 1911. Sufficient operation proved that, although a better roast and sinter could be made with the pan, and that its operating cost would probably be acceptable, its capacity on the charge provided was much less than that of the stall units already long operated. Small scale experiments indicated that the capacity of a large pan would more than equal to tonnage of the stalls. In actual operation, the mixture provided was so very much different from the ones easily made for a little cake that the results were disappoint-

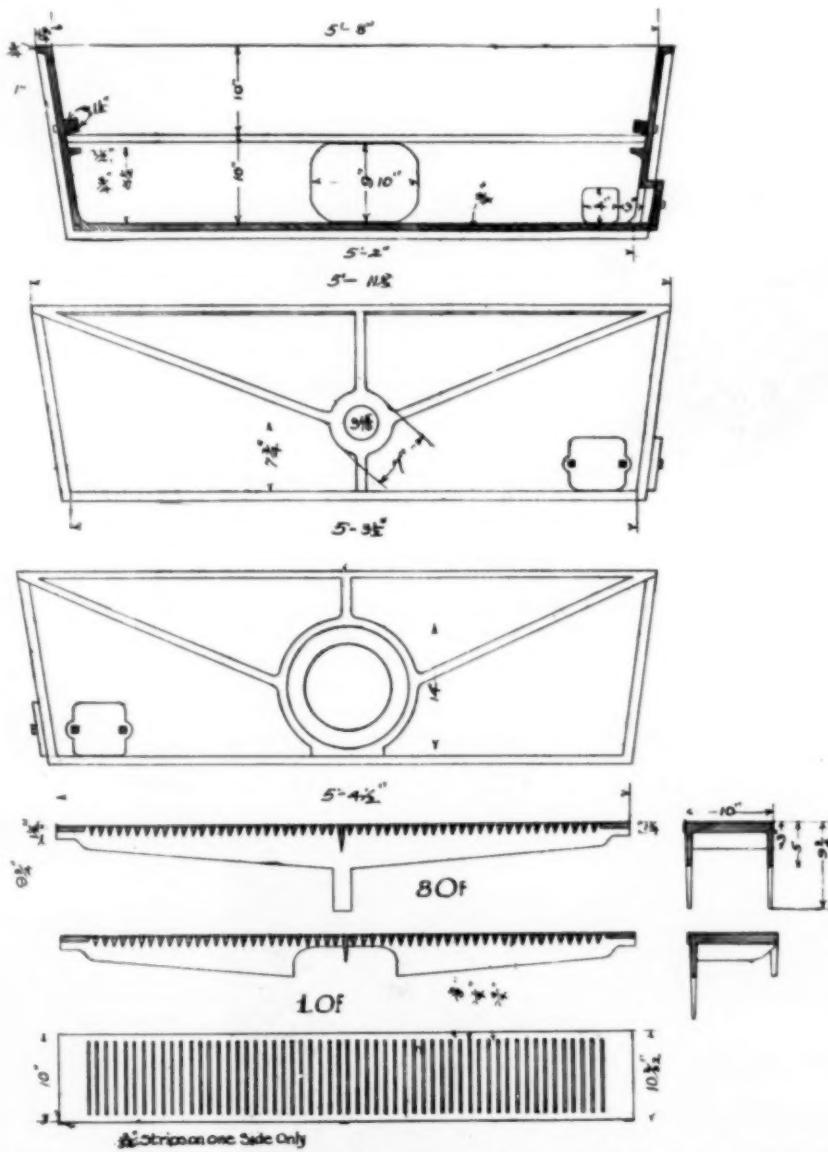


FIG. 3—SECTION AND END VIEWS, WITH DESIGN OF GRATES, OF GREENAWALT DOWN-DRAFT PAN.

ing. High sulphur charges which slag and matte with up-draft blast, merely hold their heat and refuse to finish under the influence of the suction; the low sulphur charges have no heavy ignition to do the roasting in place of the internal fuel, and leave raw patches.

After the working of over a hundred tons on the pan three of the regular stalls were slightly remodeled and connected with the fan which had served the pan; some 2,500 tons of product were then made, using suction and the regular charge. This attempt also proved disappointing in that the tonnage by no means equalled that of the regular stalls.

The down-draft system is characterized by the gentleness of

the action; there is no enormous generation of heat in an igniting bed of glowing coal and high sulphur concentrates; in fact, if glowing coal be spread on top to ignite the charge, the first burst of air through the originally porous charge is apt to so fuse the top layers that the pores are permanently closed and the rest of the desulphurisation indefinitely prolonged. The briefer the ignition, provided that the top layers are once fully lighted, so much the better will the subsequent roasting proceed.

With the gentle ignition follows a roasting and sintering equally unostentatious. Where the charge is of exactly the proper constitution there is very little motion of any of the particles; with slight contraction the cake finally becomes cold and dead. Any departure from this condition only delays instead of rushes operations. The mixture should be so uniform that each little layer will come to a glow and pass the fire on to the next, itself becoming cold.

In the fairly thin layer of all suction work the draft is from one broad free surface to another equally broad maintained at the grate. Every tendency is to close and not to open a channel for the draft. This is diametrically opposite to the tendency in all up-draft work. Accentuated by high sulphur, or too much fuel, in the mixture, the result with down-draft is some matting, but, more prominently, undue prolongation of the operation. Too much fuel in up-draft work does not necessarily so much prolong the operation as it fuses and mattes and creates blow-holes.

Those conditions which the Greenawalt pan requires are not to be considered difficult or costly of attaining. On a large scale mixing is well and cheaply done, with substances more difficult to handle than the granular ore mixture. The regulation of the moisture is not difficult to handle when the proper consistency is known; one or two per cent. from the ideal percentage is of small consequence.

#### Typical abnormal charges:

	Wt.	Cu. Ft.	Lbs.	Moist.	Pb.	S.	Fe.	Insol.
1. Too much concentrates.....	137.5	6.1	11.9	20.5	30.0	19.5		
2. Dry and fluedusty.....	137.0	4.8	16.4	19.0	26.5	16.0		
3. Too wet.....	135.0	9.0	16.0	17.2	30.1	19.7		
4. Too silicious.....	124.0	6.1	12.7	13.3	27.1	25.2		
5. Too dry and fluedusty.....	118.0	8.4	14.0	19.9	23.0	14.0		

It must be borne in mind that, although the average composition of a large batch may be exactly right it will not answer that the mixture as fed into the pan may depart from this average to any considerable extent. The average of the above charges would make an excellent mixture and sinter well; the individual charges are in no case suitable. Mere inspection is sufficient to indicate the abnormality, but the analyses tell to what exact extent. It requires some engineering to have the conditions uniformly correct.

With the dumping pan low operating costs can certainly be attained. The units require but a few minutes for charging and igniting, and then no attention until the roasting is complete. A spring is pressed back, a little push or a slight pry with a bar

and the unstable pan swings over, the cake drops out and the pan swings back, to be again caught by the head on the spring.

Moving parts or parts requiring attention are at a minimum. The pan used at Midvale had an effective bed depth of 10 inches. Two tons of the heavier mixture could be accommodated at one charge. A conservative estimate of the tonnage, based on actual operation with mixtures of mediocre suitability is that a cake of 1.8 tons roasting to 70 per cent. sulphur elimination from 15 per cent. lead and 19.0 per cent. sulphur should be dumped 7 times in 24 hours; that is a tonnage of 12.6 tons per 24 hours. On this same rough-and-ready charge either H. & H. pots or converter stalls should make an output of 16 tons per 24 hours.

With better mixtures the capacity of the pan should increase much, and it is believed that operating cost and cost of repairs and maintenance cannot be kept lower by any other type of equipment.

The cast iron pan at the Modern plant suffered no damage; the Midvale pan cracked in service, but was repaired and continued in use. Each time the pan dumped it received a severe blow, not wholly uniform, on one side, which came in contact with a wooden bumper in front of a steel beam of the floor.

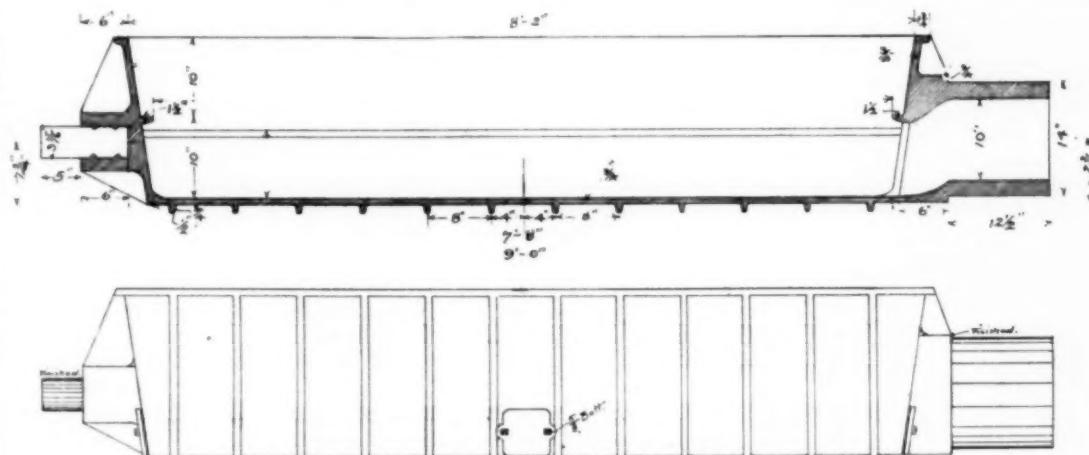


FIG. 4—LONGITUDINAL SECTION AND SIDE VIEW OF GREENAWALT DOWN-DRAFT PAN.

The analysis of the iron in the immediate vicinity of the crack indicates a good iron of the accepted quality for casting and machining, but of no particular strength and toughness.

Per Cent.

Silicon .....	2.45
Manganese .....	0.25
Phosphorus .....	0.98
Sulphur .....	0.11
Graphite .....	2.74
Total carbon.....	3.18

The particular advantages of the Greenawalt method are:

- Simplicity of operation and operating parts.
- Low cost of operation and handling material.
- Small amount of fines, from breaking cake mainly.
- Slight volatilization of lead or other values.
- Practically no flue dust, slight fume, slight SO<sub>2</sub>.
- Ideal physical structure for furnace smelting.
- Low cost of ignition, breaking product, and maintenance.

Continued experience should result that newer plants have smaller operating cost due to the better design. When we have added to this the apparently superior principles of down-draft roast-sintering there appears little hope for the older methods. Estimates of the cost of pan-roasting will be given in the sequel.

#### V. Engineering in the Roasting Department.

With the advent of blast-roasting and the establishing of more or less standard mixtures which must be fairly closely adhered to for success the attaining of low costs settles down largely

to the operation of machinery and the handling of materials. With hand reverberatory roasting the great cost was to pay for manual labor to use as a source of power. The new method dispenses almost entirely with this factor and we pay for power as generated from coal or electricity, labor cost is the price of operating the equipment, and instead of buying brick and fireclay the cost of repairs is now the price of new wearing parts and replacing corroded steel.

The engineer is called on to meet conditions of novel variety. How shall we store continually varying raw material so that it may be drawn off for sintering with its composition fairly constant, so that the coarse and fines shall always be in the same ratio? How shall we keep moist material from freezing in severe weather? How best keep moist material from drying out during hot weather? How best convey the moist and sticky charge from mixer to roaster unit? How best dump a charge from hopper to unit without segregation? What is the best method of producing high suction? These and a multitude of other questions arise when the problem comes to handle and treat the variety of material to so many processes with a limit to the capital expenditure and operating cost.

The components of the charge are to be obtained of uniform character, accurately weighed, thoroughly mixed, suitably moistened, placed in the roaster unit, sinter-roasted, discharged and removed for storage. The rest of the paper will be devoted to a brief discussion of the sequence and carrying out of these manipulations.

#### *1. Preparing the Charge.*

There is apparently only one satisfactory condition in which material may be received either at the sampling mill or at the roaster bins—dry. To accept either concentrates or ores in any other condition means greater error in original weights, moisture, sampling and distributing, besides far more difficulty in unloading.

The chief concern is naturally to be able to get the material out again and to get it out always gradually in small parcels and always of the exact mean composition whatever particular method of storage be used. To attain this the material must certainly be first uniformly unloaded. From the metallurgist's point of view it is a pitiful sight to see a sample man standing on an island of coarse concentrates in a steel car, the mud and slimes surround him on all sides and he tries to get into a small pail something which shall represent how wet it all is. What good is a sulphur determination from shovel sampling such a mess, just as certainly now, as later, we may see the ill working and high sulphurs resulting in the final roast. Aside from direct unloading, possibly using steel troughs, and hand spreading below, there is no conspicuous method applicable to unloading.

The sampling of either dry concentrates, or silicious or other ores for diluting matter, may be carried out sufficiently well for the roasting department by shovel sample at the time the cars are unloaded. Concentrating plants would certainly find it to their advantage to send out their product fairly dry and well mixed. This should add no noteworthy burden of cost to the mill, and might eliminate considerable guess-work as to their own efficiency; the smelter could well afford to allow a small bonus for the good service, or even penalize the opposite.

As to the sampling of the silicious and iron ores which are necessarily mechanically sampled in a mill, and whose fines are to be screened out for diluent, it may be said that no extra cost is justifiable for this service. A well designed and efficient mill handling several hundred tons of ore a day and sampling at a cost of 10 to 15c. per ton should do the screening for a fraction of a cent, only, additional; to crush everything to half-inch, or thereabouts, a couple cents might be added.

If raw, massive sulphides are to be crushed and roasted they should be reduced to pass a screen with  $\frac{1}{4}$ -inch mesh. The diluent need not be reduced quite as fine, but it will do no harm; nothing larger than  $\frac{1}{2}$ -inch has any place in blast-

roasters. Besides losing its intrinsic diluting effect coarser material inevitably segregates from the fine; this is very important, for the material must needs be handled several times and the un-mixing itself, without any cumulative effect, will be found bad enough.

The weighing of the constituents of the mixture presents no particular difficulty, although certain errors are to be avoided. Multiplicity of scales and the placing of scales in unfavorable locations is not only a source of serious error but costly. Standard railroad platform scales, when properly cared for and regularly tested and adjusted, will answer every purpose for making large beds or mixtures of several hundred tons. Smaller platform scales serve for weighing into tram cars or ordinary mine cars; in this case it is best to provide only one or two scales, using these much and keeping them in good condition. Great care must be taken to keep the tare of the cars correct.

A smelter weighmaster will probably return the weight of a 50-ton car of ore to within 200 lbs. of its correct weight; that is, he will probably weigh to within 1 part in 500. The usual scaler in a smelter who receives about \$2.25 a shift, when weighing out ton batches into a 3-ton tram car, will be doing splendid work if he gets his ton lots weighed to within 10 lbs. of what they actually are; this is to 1 part in 200. Besides the natural bother of weighing to mark the tare in the latter case is extremely difficult to keep adjusted. In attempting to accurately gauge small parcels of material it is more than likely that a system of measuring, indirectly referred to its weight, will be found more satisfactory. Mechanical feeding devices appear the proper thing for this work, especially for feeding from bins onto conveyor belts which will carry the material to the mixer.

Storing the supply of concentrates and crushed ores has already been touched upon; bedding the individual components in alternating layers and drawing off a vertical section which is again mixed and conveyed to the hoppers satisfies all demands and quite likely supplies as uniform a charge as may be obtained. However, in this case there is the expense of spreading and a further considerable capital outlay if arrangements are made to draw off the material into cars run beneath the beds. With uniformly fine material, such as concentrates, either flat or hopper bottomed bins might be used, but without spreading, segregation almost certainly results, and they are difficult to draw without again involving the same difficulty. A variety of methods of storing the materials might be suggested besides these more common ones; particular attention must be given for the abnormal conditions of wetness and severe freezing weather. Besides promptly doubling the cost of this part of the work, freezing not only may greatly decrease the tonnage but the charge cannot be mixed, it roasts poorly and the product is bad.

If the material is assembled in beds it may be drawn and sent directly to the mechanical mixer. If the material is drawn from separate bins it may be first assembled into cars and transferred to the mixer, or run onto conveyor belts or onto the same belt and delivered to the mixer. For existing plants this item of cost is usually rather high; in these plants the roasting department has often been built after the plant was laid out; in planning for new plants or rebuilding old departments foresight may easily save a few cents per ton by happy combination of method and arrangement of parts.

The mixing of the charge has already been referred to as of vital importance; hand mixing is not only costly and inefficient but does not do the work. Of the mechanical mixers available in the market the batch mixers probably give the best results for the reason that the material may be retained sufficiently long for thorough incorporation. If any continuous mixer should accomplish the same it would be available. Concrete mixers are handy and reliable, while for large tonnages other designs might serve better, the points of importance are that the mixing shall be complete, the batch must be visible so

that its consistency may be seen and more water added if necessary, it is best if the stuff is not only tumbled but incorporated with blades or baffles, the machine should not choke easily and as little power as possible required. If much lumpy slimes or fluedust or baghouse dust is to be worked in, the short tumbling and small cubic content of ordinary concrete mixers is hardly sufficient.

Pre-roasting is certainly an important consideration in the preparation of mixtures for the H. & H. process. That it can still survive and in conjunction with the roasting pots produce a product at such a low cost speaks well for its simplicity and low operating cost; indeed, a man to bring up the ore, another to care for the roast and one to fire is the full complement for a battery of several. It is an expensive method of preparing the primer for a pot but it also reduces the amount of diluent which would otherwise be required. This is possibly desirable with up-blast work where some ores are too rich and better briquetted but hardly offers any incentive with down-draft installations.

Depending on conditions, the cost of pre-roasting will be from 25c. as given by Huntington and Heberlein, to 63c., as given by Ingalls. Tonnage and kind of material are the great factors.

Considered in its ultimate economy, pre-roasting of pyritic-ferous ores is on about the same basis as crushing coarse iron ore and adding it for diluent in the roasting units; either method produces a sinter high in iron which will be balanced by new silicious material at the blast furnaces. To add the diluent in the shape of screenings of both silicious and irony material, thereby ridding the furnace burden of fines and putting all concentrates, fine material, flue dust and baghouse dust into porous sinter promises the lowest total smelting cost and best recoveries.

Allowing 5c. a ton as expense of spreading in beds and 3 men to assemble, mix and transfer to the units, with 2c. per ton added for power, the cost of preparing the charge is:

Bedding .....	\$0.05
3 men @ \$2.00.....	0.06
Power .....	0.02
<hr/>	
\$0.13 per 1 ton	

provided 100 tons are handled each shift. Supposing the original design and character of materials supplied are both good, it does not seem impossible that 2 men and the same power should do the work eminently well. This is, then, an estimated cost of 6c. a ton for preparing the charge.

## 2. The Roasting Operation.

The charge once prepared and assembled over the roaster unit those operations pertaining to the roasting act itself may be said to begin. As commonly carried out, these separate steps comprise this duty: charging the units, igniting, blowing or sintering, and emptying or discharging.

After the assembling and mixing and moistening of the charge it has quite a sticky consistency, the proper consistency for it to best sinter is exactly that at which it flows through an opening or in a trough with most difficulty; in other words, that at which it best retains any given shape. From this it follows that we must not expect to put it in a large hopper bin over the units and be able to draw it without considerable difficulty; besides the possible segregating we now have a mass which will not pack easily nor flow easily; in fact, it is just what we want and we must merely adapt conditions to handle it properly. Anything more than a small, simple and wide-mouthed hopper directly over the unit can have only two advantages; one is as storage to equalize irregular supply, the other, that storage disseminates the moisture very nicely and uniformly; both of these advantages, to my mind, do not compensate for the disadvantages involved. Any cars or hoppers handling prepared charges will thus be steep-sided and with wide mouths. If conveyor belts are used, ample provision is

made to brush or to scrape the belt directly after it discharges.

If the charge is dropped upon the grate of the unit, this already made ready with protecting bed or burning igniting material, the fall is broken with appropriate strips of plank or steel; it is quite as readily slid down a wide and steep trough. Once in the unit, the sticks are removed and the material spread with hoe or shovel; suction work requires an additional smoothing with a rake; any coarse bits on top are also removed, as they hinder ignition.

The igniting has already been quite fully described for all types in the literature, and, with proper preparation of the charge, presents no particular difficulties. The ignition with oil or suction equipment is likely the cheapest of any method; this need not exceed a cost of 5c. per ton, which is a lower figure than we are aware of as otherwise attainable. For use in the fine nozzles the oil must be free from grit and scale and the fittings well made.

The sintering follows fast and well if the previous work has been well done; nothing will avail in the units to compensate for poor preparation. With any type of unit the blast or suction is turned on and results awaited.

The factors of moisture, fineness, uniformity, sulphur, lead, iron and silica in the mixture have been pretty thoroughly discussed and their limits defined. More than already stated, it is to be noted that some zinc is usual in all ores and concentrates; up to 3 or 5 per cent. its effect is certainly not more harmful than possibly giving higher sulphur in the product. Sulphide ores containing more than this amount should be milled for its separation. Flue dust well incorporated into the mixture does no particular harm even if it does contain large amounts of arsenic.

The blast regulation at the unit is more to compensate for faults elsewhere than as belonging to this detail of sintering, except with up-draft pots or stalls where the blast is cased or shut off while the blow-holes are punched. The sulphur, arsenic and lead fume from suction units will accumulate in a fan placed near the unit; the fan is thus to be made with the case split horizontal and hinged so that it can be quickly opened and cleaned. A large fan placed at considerable distance from the units and supplying several with suction will have slight inconvenience from the small amount of fume made. A few pounds of the sticky fume will put a small although powerful fan entirely out of commission; in a large flue this might collect for months before requiring removal.

When the operation of roasting and sintering is completed or has been carried as far as practical, the discharging takes place according to the particular equipment installed; any particular cost for this part of the work should be eliminated. Thus, at Midvale, the services of the ram man and the crane chaser are practically devoted entirely to this detail, involving a direct cost of 4.5c. per ton. With the Greenawalt pan, which is in unstable equilibrium when loaded, a slight push inverts it and the cake falls out, breaking on the angle irons below and sliding off into the steel boat or skip placed below, ready to be taken to the breaker.

The Dwight-Lloyd machines are supposed to work equally simple and with the same minimum cost.

Nothing more complicated or costly of operation will be tolerated in future design; even for these simplest cases one must bear in mind the properties of the material and use no material or construction which would be quickly damaged by the hard, rough and hot cakes in their entirety if, perchance, they should not be broken as expected.

Referring to the data already given, the labor cost of the roasting operation is given as 20.7c. at one plant, and for the other plant we may likewise find a sum of similar items which will amount to 6.3c. For a Greenawalt pan we may properly estimate that 2 men, whose pay shall equal \$4.00 an 8-hour shift, will be able to dump and load a pan every 15 min., or handling a tonnage of 60 tons per 8 hours their labor will cost 6.7c. per ton. If we add to this the services of another man

to clean grates and the fines from beneath the grates we shall have a cost for labor of approximately 10c. per ton. This item of operating cost for the Dwight-Lloyd machines might be supposed to be somewhat less.

To furnish the blast or suction requires more power than the sum of all the other power requirements in the roasting department.

If, let us say, 100 continuous horse-power is required to supply air to treat 300 tons of material per 24 hours at a cost of \$30.00 per horse-power year the cost is very nearly 3c. per ton.

Depending upon the cost of coal or of the electricity, this item is in so much a variable factor; the price of coal at most lead smelters, due to the long haul, will hardly be at a low figure and the price of power will probably be much greater than in this last instance. In the case of two plants already considered the cost of power has been estimated at 9 and 10c. per ton of product, respectively. Without further explicit data the cost of a suction plant will be assumed at this same figure (9c.) per ton.

### 3. Crushing and Loading.

The employment of hand labor, alone, to break and load the product of the roasters is altogether too expensive to be considered where the amount of product is as large as is likely to be found in any permanent lead plant. A certain development in learning how to treat the roast mechanically is to be recorded, but, the usual cost of this operation is evidently far greater than will long be tolerated.

Allowing a crane man, a hoist man and 3 laborers as at work on this detail to care for 90 tons a shift the labor cost is about 11c. a ton. The question arises, should not a hoist man and one laborer be sufficient, and we inevitably answer that we believe the plant could easily be so designed that with not more than present power consumption this amount of labor can do the work. The tough, hard, massive and hot cakes commonly made with up-blast must first be broken before going to the crusher; even then an inordinate amount of labor is consumed in pounding, prying and turning about the big pieces before they settle into the jaws of the crusher. To use a Blake-type crusher of larger size than 24 in. x 36 in., which is a common size, is evidently not practical; it is, however, not so difficult to imagine a set of slow-turning and massive rolls above the crusher, whose teeth should engage and draw in the cake to be broken and passed down to the crusher for final sizing.

It ought not to be too difficult to plan and construct units so that a skip could catch the finished cake and carry it directly along and up and over the crushing equipment, dumping it and returning for the next load. Meanwhile, we suppose the cake to have fallen in, been crushed, and is now dropping down into the railroad cars which enter directly beneath. We thus come to estimate a cost of only 5c. per ton, which we presume will more than likely be bettered in the plants of the future. In particular with down-draft work, which produces a much more open sinter than up-draft, this estimate will be reasonable enough.

In regard to handling this type of material with elevators and conveyor belts, or conveying steel buckets, the burden of proving adaptability is still evidently on the apparatus. Skips are proving their usefulness in many instances, and the cranes also in use testify for their high cost for up-keep.

### VI. Summary of Operating Cost.

Various summaries of operating cost for different plants are at hand in the literature indicated.

In his paper in the Engineering and Mining Journal Herbert Haas supplies the following estimate for a plant consisting of 20 of his units, each of which is to handle 20 tons per 24 hours. To the table he adds the cost of bedding, mixing and conveying the material, which items he says may amount to from 5 to 20c. per ton.

### COST PER TON OF SINTERING WITH 400-TON PLANT.

#### Fuel and Labor.

Fuel for primer, 1 per cent. on charge, @ \$2.50 per tonne..	\$0.025
Six men for priming, two per shift, @ \$2.00.....	0.03
One and one-half man per unit, @ \$2.00.....	0.15
Six crane men, @ \$2.50,	
Three crane operators, @ \$3.00 } .....	0.06
Three foremen, @ \$4.00.....	0.03
Six laborers, @ \$2.00.....	0.03

#### Power.

50 hp-hours per unit for air @ 1c.....	0.025
800 hp-hours for crushing sinter @ 1c.....	0.02
600 hp-hours for operating crane @ 1c.....	0.015
Crusher repairs, wear and tear, supplies, etc.....	0.03
Repair of plant, wear and tear, supplies, etc.....	0.10
Amortization: 10 per cent. on \$8,000 per unit.....	0.11
Interest on invested capital, 5 per cent. on \$8,000 per unit.	0.055

Total estimated cost of sintering per ton.....\$0.680

Ingalls, in 1906, estimated the cost of pre-roasting in the H. & H. process to be 63c. per ton, estimating details as follows:

Labor: 3 men at \$2.50.....	\$7.50
Coal: 18 tons at \$2.00.....	36.00
Power .....	3.35
Repairs .....	3.35

Total .....\$50.20 for 80 tons, or 63c. per ton

He estimates the cost of converting thus:

Labor: 3 foremen at \$3.20....	\$9.60
9 men at \$2.50.....	22.50
Power: 21 hp. at 30c.....	6.30
Supplies, repairs and renewals.	5.00

Total .....\$43.40 = 60c. per ton of char

Total cost of treatment Ingalls also gives as follows:

Crushing, 1.0 ton at 10c.....	\$0.10
Mixing, 1.0 ton at 10c.....	0.10
Roasting, 1.0 ton at 63c.....	0.63
Delivering, 1.1 tons to converters at 12c.....	0.13
Converting, 1.1 tons at 60c.....	0.66
Breaking, 0.9 ton at 60c.....	0.54

Total .....\$2.16

Ingalls also remarks that the Sävlesberg process will probably be a serious competitor insomuch as the roasting is omitted, thus making the process cost 63c. less.

In the same paper the cost of the Carmichael-Bradford process is given to be as estimated by the patent owners:

Labor reckoned at \$1.80 per 8 hours, gypsum to cost \$2.40 per 2,240 lb., and coal at \$8.40 per 2,240 lb.

0.25 ton of gypsum.....	\$0.60
Dehydrating and granulating gypsum.....	0.48
Drying mixture of ore and gypsum.....	0.12
Converting .....	0.24
Spalling sintered material.....	0.12
0.01 ton coal.....	0.08

Total .....\$1.64

In 1910 Hofman gave the cost of the blast-roasting at Midvale, Utah, as \$1.25 per ton when treating 320 tons a day. He also says that the estimate of Ingalls of \$2.16 per net ton of blown mixture should be deducted by one-third for Rocky Mountain plants and by one-half for the Mississippi Valley region. This would leave his estimates \$1.44 for the first and \$1.08 for the second case.

In this same paper Hofman states that at Salida, Colo., a Dwight-Lloyd machine is treating 50 tons of mixture a day at a cost of 75c. per ton, also that two machines in Illinois are treating 100 tons a day at a cost of less than 50c per ton.

In 1908 Austin, in the *Mining and Scientific Press*, estimated the cost for a 50-ton H. & H. plant to be:

13 men at \$2.50.....	\$27.34
1 foreman (part).....	1.40
Fuel, soft coal, at \$1.55.....	9.30
Power, est.....	3.00
Light, est.....	0.40
Repairs and supplies.....	2.50

Total ..... \$43.94 and compares this at a cost of \$1.02 per ton (the product weighing 15 per cent less than the raw ore) with a cost of \$2.60 for hand roasting and briquetting the same material.

Huntington and Heberlein, in their paper, conclude that pre-roasting cannot cost more than 25c. a ton, and that, in the United States, where pre-roasting is more desirable than in Europe, it will cost 24c. a ton less than direct blowing.

In this discussion the cost of H. & H. treatment has been given as:

200 ton capacity.....	85c. per ton
same plant, 400 " " .....	69c. " "

and converter stall treatment as

270 ton capacity.....	78c. per ton
same plant, 350 " " .....	68c. " "

In each case this refers to per ton product and to operating cost only.

To summarize for a down-draft plant, in particular for a Greenawalt pan installation, with even less than customary capital outlay, operating cost:

300 tons capacity—	
Preparing charge .....	\$0.06
Roasting .....	0.10
Crushing and loading .....	0.04
Limestone to cover grate.....	0.02
Igniting .....	0.05
Power .....	0.12
Supplies .....	0.02
Repairs .....	0.05

Total .....	\$0.46 per ton of product
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Armour Institute of Technology,

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### The Teaching of Technical German in Engineering Schools.

BY FREDERICK WM. SCHOLZ.

Considerable attention has been paid within recent years to the teaching of engineering subjects in technical schools with the idea of obtaining greater efficiency and giving the students the best instruction. Very little has, however, been done in the matter of teaching modern languages to engineering students, so that at this time that phase of the technical curriculum is neglected in most engineering schools. It is the purpose of this article to enter into the questions involved in the teaching of modern languages, particularly German, to engineering students, and to suggest methods of solving this important problem.

The first question which we have to discuss is the need of foreign languages, especially German, to the engineer. The rapid and phenomenal progress of engineering in America and Europe has resulted in the publication of a vast amount of literature, published in form of periodicals, textbooks and pamphlets. This covers every field of human achievement, and comes from America, England, France, Germany and Italy, principally. While the literature of England gives us no trouble linguistically, that of France and Germany offers quite a different proposition.

German engineering literature, especially, is among the most valuable published in any country, not excepting the United

States and England. To prove this assertion let us take a short glance at German achievement in the engineering field. The Königliche Materialprüfungsamt at Gross Licherfelde is acknowledged to be the best testing station in the world. The Zeitschrift des Vereins Deutscher Ingenieure, published by the German mechanical engineering society stands foremost in the ranks of engineering periodicals. No society publishes such extensive and accurate papers on tests and investigations on engineering subjects, as this one in the reports called "Mitteilungen über Forschungsarbeiten." Similar conditions hold true in the field of electrical engineering, mining and chemistry. An important point in the wonderful industrial advance made by Germany is the support given to research and investigation by the government, the universities, engineering schools and business corporations. The professors of engineering at the schools, the engineers in the service of the great corporations are encouraged and aided by the government and engineering societies to engage in research work, and as a result we have German engineering literature filled with the valuable results obtained by these men.

Another point which makes German such an important language is the fact that it is not only the language of Germany, but also that of Austria, Switzerland and even non-Germanic countries like Belgium and the Netherlands. The important engineering works of France, Belgium, Italy, in fact of all of Continental Europe, are usually translated first into German, and any large engineering project is bound to have one or more German engineers among the staff, who report the progress of the work in the German engineering press. The result is that the German engineering literature is an indicator not only of the work in Germany alone, but practically takes in all of Europe, so that the engineer, able to follow the German technical literature, keeps abreast of all the important projects going on in the whole of Continental Europe.

Very little of this literature is, however, translated into English and thus brought to the attention of our engineers. That the material is important is recognized by American periodicals, for many of them have foreign departments devoted to the abstracting and translation of foreign engineering news. The really important material, however, does not find its way into American papers, for there is so much material of value published in our own language that space for foreign news is limited. It cannot be claimed that all the good foreign material is published in our papers, for the facts do not bear out the truth of this assertion. Moreover, limitations of space permit abstracts only to be published, so that the engineer who wishes to get detailed information must go to the original source for the full report. This presupposes that he is able to read the technical German with some degree of fluency, which, unfortunately, is true only with a very small percentage of engineers. If the article is important enough to him he can have it translated at some expense and at the risk of an inaccurate translation, or else he has to do without the information.

The fault lies not, however, in most cases, with the engineer, but rather with the lack of training in modern languages in the engineering school. While some schools teach German and French, they teach it in such a manner that the student obtains no benefit from the course and is glad to be relieved of further study in that field. Some schools do not teach modern languages at all and rely on the preparatory schools to furnish the necessary equipment. As a result the foreign engineering literature, especially German, is a closed book to the engineering student, and he has to rely on secondary sources to obtain his information. The remedy lies plainly at the door of the school, and we now come to the methods to be used in teaching technical German in the engineering school.

The first point is the ultimate end to be obtained. The final aim is to enable the coming engineer to read a German technical paper with such a degree of fluency that he can determine whether the article in question is suitable for his purpose and then obtain the desired information from the same.

The aim is not to teach the man how to speak German, for comparatively few of them go abroad; and if they do, they will learn more German in one year than they will learn in school in four years. It is, therefore, false to apply the methods used to teach men how to speak German to the teaching of engineering students. Two or three years of this type of instruction may, indeed, give them some slight command of the language and a little reading ability, but it will not teach them the technical literature and the vocabulary they need to read this. The course should be arranged in such a way that the student, after two or three years of German instruction, is able to read any technical paper in his particular line with some speed and with a slight use of the dictionary.

The next question is the accomplishment of this purpose. If we are able to arouse the interest of the student and show him the usefulness of the foreign language we have achieved quite a little, for if the student is interested in his work he will make an effort to advance. If he sees that the language is a tool which is useful not only in the far-off future, but at the present time, he will overcome the dislike shown by most engineering students to languages and apply himself to the task set before him.

The first step in this direction is the use of proper textbooks in the teaching of the language. Comparatively little grammar is needed for the reading of technical German, and a half year or one year, at the most, of grammar will be sufficient to meet the requirements. The grammar should not be taught separately, however, but should accompany the reading of technical literature, running parallel with the latter, till the students are thoroughly familiar with the necessary grammatical constructions. A short German grammar of good standing will suit the purpose.

The more important problem is that of the reading matter. There are various scientific readers on the market which are used in some of the schools teaching languages to the engineering student. None of these meet the need, however. First of all, they are only makeshifts, made up of articles picked up here and there; and, secondly, they do not, as a rule, use the language of the engineer, but rather that of the physicist, the theoretical chemist, the mathematician, whose language is not that of engineering literature. The student, moreover, sees no difference between this textbook and any other German reader and is likely to do only grudging work with the same.

The best method would seem to be the reading of the current engineering journals published in Germany and sent to the engineering school from abroad. Take the current engineering journals published in the various branches of engineering, place them in the hands of the students and their interest will be immediately aroused, for here they have the actual literature of the country and the information they get is that of the practical engineers at work in the field. Of course, careful selection must be made in the periodical literature, and easy reading matter, which can be found without difficulty, should be read before the more difficult theoretical material is attacked. The idea is, then, to start the reading of technical papers in the first year, beginning with the easier papers and gradually working up to the more difficult ones.

The objection might be raised that before the student is able to read the papers he should first get some sort of a working vocabulary by reading easy, common-day experience stories. We must remember, however, that technical literature is obliged to use the common, every-day words just like the other reading matter, and that the student will get this vocabulary along with the technical expressions. If we read other extraneous matter first, we will give him quite a number of useless words, which he will have to discard and forget when he comes to the reading of technical literature. Why not, then, give him the right literature at the start and have him get that vocabulary at once, which he will use all his life in the pursuit of his profession and from which he will not need to drop any words as useless.

After two years of technical German reading, giving three periods a week, the classes should be separated into several divisions, depending on the line of work the men intend to pursue in later life. The first two years should see the reading of engineering literature from electrical, mechanical, civil and some mining engineering, as also metallurgy and industrial chemistry. This would furnish the men with a vocabulary of words which occur in all of the branches of engineering, for mechanical, electrical, civil and mining engineering are, after all, closely connected in some respects, and the terms used in one line occur in another line. In the third year, after the classes have been divided into sections, each section should read those technical papers which are closely allied to their own line of future work. For example, the mining engineer might read *Stahl und Eisen*, *Glückauf*, *Bergbau*; the electrical engineer might read *Elektrotechnische Zeitschrift*, *Elektrotechnik und Maschinenbau*, etc. This would give the men a larger working vocabulary in their own line of work, so that at the end of the course they have a stock of words applying to general engineering and a special stock in their own line.

The work might also be made more interesting by procuring large blueprints of gas engines, turbines, valves or other machine parts and having a discussion on the constructive features, using German terms for the various parts. The use of technical dictionaries should also be encouraged. The series of six-language dictionaries of the Deinhardt and Schlemann type, covering the various branches of engineering, would furnish excellent working tools in making translations. Moreover, their usefulness would not cease after the completion of the language course, as they could be used by the men as an aid in reading the technical papers in their professional work.

The next problem is that of the instructor in technical German. Here we have to deal with a very difficult proposition, because little attention has been paid to this phase, and therefore hardly any provisions have been made to equip men for this work. We have three types of men for consideration, namely, the engineer, knowing German and English; the classical student, knowing both languages but with no interest in engineering, and the man who has command of both languages and understands something of engineering.

Take the first man, who is a trained engineer with a command of both languages. If he is going to enter the teaching profession he will likely do well to teach the engineering subject in which he is most proficient rather than the language which to him will be a secondary matter. He is, therefore, out of the question for our purpose, unless he can be induced to take up the teaching of the engineering language instead of his engineering subject. If this can be done, he would be the best man for the work, for not only does he know both languages thoroughly, but he is able to determine the right terms in case of difference between German and American terminology.

The second man is the classical student in command of both languages, but with no interest and knowledge of technology. While he may be splendidly equipped for the teaching of German literature, it is very doubtful whether he would be the right man for the teaching of technical German. First of all, he has probably no interest in engineering, and no man will do good work in a line in which he has no interest. Secondly, his vocabulary will be entirely different from that of the engineer, so that he would not have the linguistic equipment to take up this branch of teaching. Again, if he knows nothing of engineering practice and does not follow up engineering literature, he will have to rely for his translation on dictionaries entirely, which are at times inaccurate, and accuracy is one of the greatest essentials of engineering translation. If he undertakes to translate something about which he understands nothing, he cannot do justice to the work; he will only make himself ridiculous in the eyes of the students and will fail to obtain results.

It is, therefore, necessary to have men who know something about engineering, who are interested in the subject and

have complete command of both English and German. The instructor need not know how to derive a complex formula in electrical engineering, but he should know something of the practical end of engineering and be thoroughly familiar with the manufacturing processes, machinery, mechanical devices, in fact, with all the subjects with which he deals in his work. Extensive reading, a thorough command of both languages and some technical training are the essential requirements.

Doubtless there are such men with the proper training who could take up the work with success. But the constant neglect of this branch of engineering education has not brought them to the front so that we do not know at present how numerous they are and where we are to look for them. Create a demand, however, and show that in this line there is room for pioneer work for men who are willing to devote themselves to the successful solution of this important problem, and the right men will step out and take up the task. Then, when the work has been advanced to an efficient basis, introduce courses in the university curriculum, where men can be expressly trained for the purpose of teaching technical languages to engineering students. Place the man with the right linguistic equipment, with some knowledge and interest in engineering subjects, and devoted to his profession as teacher in charge of the language classes in the engineering schools, and he will transmit his enthusiasm to the students and will obtain results which will be of practical value to the student and an inspiration to the instructor.

Finally we come to the question of the amount of time to be devoted to the study of languages in the technical schools. It is a fact that the schedule of the engineering student is so heavy that it is very difficult to push in any more work. Yet, in view of the final results which would be accomplished with proper methods and the value which the future engineer will derive from the study of technical German, it seems that there ought to be room in the training of the engineer in the linguistic field.

If the time could be spared it would be well to give three years of German, divided into two years of three periods a week, and one year of two periods a week. Of course, the ideal would be four years of German, divided into the above divisions with one period a week for the last year. But if this is impossible, three years will meet the purpose to a considerable extent. This presupposes that the student had no German in the preparatory school and, of course, this makes his work very difficult. The best method would be to have the engineering schools demand two or three years of German as entrance condition, and then go on and build upon the foundations laid by the preparatory school and equip the engineering student in the special line of technical German. The fact remains, however, that the need to our engineers of a foreign language, particularly German, becomes greater each year, and that any step which will tend to bring the teaching of modern languages in engineering schools to an efficient basis will be a long step in the right direction.

### Progress in Mill Design.\*

BY FRANK E. SHEPARD.

Twenty years ago the Wilfley table was not known, any form of pulp classification was considered too scientific; and the cyanide solution as a solvent for gold and silver formed an interesting experiment only under laboratory conditions. But when the Cripple Creek dump ore, worth about \$3.25 per ton, is treated by improved systems of milling for about \$1.25 per ton, and 2 per cent porphyry copper ores of Utah and Arizona are treated with recoveries of from 70 to 75 per cent, it is time that greater attention be given the low-grade ores of

Colorado, the great problem now before us. A generation ago recoveries of 50 to 60 per cent were considered good enough; to-day we must think of saving in the nineties.

Lower grades of ores must of necessity call for more complicated mill systems, and while the manufacturer is constantly striving to simplify the machinery, the mill superintendent and his men must realize that greater problems demand greater intelligence and better practical skill. If we are to accomplish these high rates of recovery there must be good team work and every man must be a friend to the system.

Consider, then, some of the methods and mechanical details by which we may raise the efficiency of our mill systems.

#### Hand Sorting of Ores.

In the mill systems of Joplin, Mo., it is customary to crush the entire lot of mill ore which, in the sheet-ground districts, is composed of about 95 per cent flint gangue and about 5 per cent lead-zinc mineral. The flint gangue is very hard and in the crushing great quantities of the galena and blonde must be reduced to slime with consequent loss in the mill system.

Here would seem to be a most favorable opportunity to produce higher recoveries by means of hand sorting. If the original mine ore be sorted and one-half the gangue thrown away, the sorted ore delivered to the mill bin would contain about 10 per cent mineral instead of 5 per cent in the mine run, and the volume to be treated in the mill would be reduced nearly one-half.

Much of our Colorado ore is finely disseminated and may not admit of hand sorting, but whenever possible this method of concentration should be carefully considered. Excellent designs of picking belts are furnished by a number of manufacturers and a well-designed plant may be automatic as far as the conveying of the material from mine bin to mill bin or the dump is concerned. Coal companies find it profitable to hand-pick the slate from the coal and there should be equal opportunity with the ores of precious metals. The women and children of Mexico are well known as skilful ore sorters and the patio of nearly every Mexican mine includes this branch of concentration.

#### Coarse Crushing.

In large mills having capacities of thousands of tons in twenty-four hours the gyratory crusher appears to be in general use on account of the large receiving-hopper capacity, the automatic feed and the absence of vibration. For smaller mills the Blake jaw-crusher is in almost universal favor owing to simplicity of its mechanism and the few parts to be renewed. An interesting development of the gyratory principle is found in the Symons gyratory crusher, which requires only one-half the vertical height of the usual gyratory design. Crusher-jaw plates of manganese, chrome and hardened steel are rapidly taking the place of the hard cast-iron or even cast-steel.

Under certain conditions the coarse crushing plant may be located at the mine. Electric power removes the obstacle of the high fuel cost at a remote station, and the reduced size of ore may be an important factor in loading and conveying details. With the possibility of hand-sorting the ore and eliminating the waste rock at the mine, the capacity of the tram or conveying apparatus could be reduced.

#### Uniformity of Mill Ore.

One of the most serious mistakes in the storage of mill ore is found in any plan which will not insure a uniform mixture of coarse and fine ore. In connection with the grizzly above the crusher for screening out the fine ore, there should be installed under the grizzly a chute so placed as to deliver the fine ore falling through the grizzly to a point under the discharge from the crusher, so that coarse ore from the crusher shall be evenly mixed with the fine from the undersize of the grizzly. Then the day shift will have the same ore mixture as the night shift and the mill should run steadily with little change of adjustments.

\*Excerpt from an address delivered at Telluride, Colo., under the auspices of the Colorado Scientific Society, Colorado School of Mines, and Telluride Club.

With problems of a large tonnage and with ores fluctuating in grade as well as physical characteristics we may study with profit the ore bedding system in large smelting plants. The handling of large quantities of ore by automatic belt conveyors and crane systems, from suitably mixed beds of ore, might result in improved mill recoveries sufficient to warrant the increased investment.

#### Automatic Sampling.

The basis of all careful mill work is correct sampling, and sampling to be correct must be automatic. Messrs. Taylor and Brunton were pioneers in correct sampling methods and in their plants a system is followed of taking a one-fifth cut, re-crushing this one-fifth sample, then taking another one-fifth cut of the recrushed sample, and so on—the whole sampling plant including five oscillating time samplers and crushing rolls between each sampler. In large milling systems a separate and automatic sampling plant on the lines indicated is well warranted, but in smaller mill systems the plan must be simpler. Two automatic samplers cutting a fifth or tenth sample, with a mixing cylinder between the two, will give a sample one-twenty-fifth or one-one-hundredth of the entire mill feed.

Under some conditions this sample must be taken after the ore has passed the grizzly and crusher; then it may be necessary to instal an extra elevator to give sufficient fall for subsequent mill operations. At the expense of additional installation it is certainly worth while to know the value of the ore with which we start operations. In a roll crushing plant a good position for the automatic sampling plant is just below the discharge spout of the elevator delivering the product from the coarse rolls, provided that no other than the original ore stream enters the coarse rolls.

It is even well to consider sampling the various feeds to the different mill units so that every phase of the mill system may be studied. Samplers have been designed to be controlled electrically from a master clock, so that the samples may be taken at regular intervals, or at times not anticipated by the mill operators. Such a system need not be too complicated and with reasonable care will give most valuable records.

About twenty years ago Mr. J. H. Ernest Waters, of Telluride, became interested, while in Denver, in a steam-pressure recording gauge, purchased one and had the gauge installed in his office and connected with the steam mains of the boiler plant, without arousing the suspicion of the firemen. He took records for a few days and obtained some charts with steam-pressure lines looking similar to the trail leading to Bridal Veil Basin. Then he explained the recording gauge to the firemen and showed them the records, and then followed records with a nearer approach to the circle. No doubt that pressure gauge paid for itself many times in coal saved and a more uniform steam pressure. This to illustrate the value of accurate records.

#### Fine Crushing.

Now we come to the parting of the ways, with advocates asking us to take various paths. At this stage of mill operation we find, in Western mills, three general systems of fine crushing:

Rolls.

Stamp batteries.

Roller mills.

For the reduction of ore from crusher product,  $1\frac{1}{2}$  in. or 2 in. size, down to a suitable size for coarse jig or coarse table concentration, the modern crushing roll employs the correct principle for the liberation of the mineral crystal with the least amount of slime product.

Modern crushing rolls are provided with adjustments so that the space between the shells may be changed while the rolls are in operation. Other details provide an end adjustment of the shafts to prevent beads forming on the edges of the shells. Forged steel shells are the best for modern conditions. This material may be machine-finished with special steel tools

and the surface of the shells thus maintained true for fine crushing.

Modern crushing rolls, with machine-finished forged-steel shells, are used to reduce ores to a product as fine as 30 to 40 mesh, and if reasonable care be given the shells such fine crushing is possible. This means an even and regular feed across the entire face of the roll shell, and a new shell for the fine rolls whenever grooving or uneven wear appears, the worn shell of the fine roll being transferred to a coarser crushing roll. In large plants having shops a lathe may be provided to resurface the worn shells.

The tendency, however, is to use crushing rolls for products not finer than about 12 mesh, No. 19 wire, equal to 0.042 in. This 12-mesh product is about the maximum for concentrating table feed, and with the short tube mill already being established as an economical system for fine crushing, the range of roll crushing does not need to be outside the limits of, say, 2 in. maximum down to  $1\frac{1}{4}$  in. minimum.

For the recrushing of jig middling rigid rolls have advantages over the spring rolls. The average size of the mineral crystal imbedded in the middling particle determines the minimum size to which the particle shall be crushed. The rigid rolls may be set to the desired size and the middling particle is crushed down to this size, and no further, the aim being to break the gangue portion away from the mineral crystal. The use of the rigid roll insures the maximum size of mineral crystal with the least production of slime.

One of the great surprises in the development of mining machinery is the strong position occupied by the stamp mill. Here is a most complicated array of mechanism, producing a chaos of noise, requiring elaborate foundations, and a large number of renewable parts, and yet the ring of the stamp goes merrily on through methods of amalgamation and lixiviation, then concentration and on to cyanide.

South African practice deals in terms of 2000-lb. stamps, and the end is not yet. Such heavy stamp weights call for massive foundations and great increase in the dimensions of mechanical parts and yet the practice appears to be commercially successful.

The use of concrete in mortar blocks is a good advance in point of construction, maintenance and capacity. Cast-steel for shoes, dies, cams, tappets and boss heads shows increased service over the former cast-iron parts. The narrowing of the mortars at the issue-level results in greatly increased capacities. The gradual change from inside to outside amalgamation reduces labor costs and loss through shut downs. Separate motor drives for battery units simplify the plan from the complicated jackshaft drives. Specifications now call for chrome-nickel steel as a material for the camshafts.

Structural steel for the battery frame is quite possible. Probably the best form of battery post is a box column filled with concrete to reduce the sound and vibrations. Designs have already been made for concrete battery posts reinforced with steel rods molded integral with the concrete mortar blocks. Sub-bases weighing about 7000 lb. give sufficient bearing area for the mortars on concrete blocks, instead of anvil blocks weighing 16,000 to 20,000 lb.

The large mills treating the porphyry copper ores have brought about a rapid development in roller mills, more familiarly known as high-speed Chilian mills. In crushing tens of thousands of ore a day certain merits must be found in this type of intermediate crushing device, but when one considers the renewal of large tires and die rings, screens, bearings, pinions and gears the query appears will this type of crushing mill endure in mills of great capacity?

#### Ball Mills.

The cement industry has given us the tube mill, that fine crushing device of great simplicity, and we shall yet appreciate the great value of the ball mill as a desirable intermediate crushing device, possibly supplanting roll, stamp mills and high-speed Chilian mills.

The Krupp type of ball mill is successfully used as an intermediate crusher between the coarse crushing mill and the tube mills, which grind to a final product of 100 to 200 mesh. The Krupp ball mill is cylindrical and is rotated on trunnion bearings, the feed being introduced to the inside of the cylinder at one of the trunnions.

On the inside of the cylindrical shell is arranged a series of steel shelves and perforated plates. On the outside of the cylindrical shell is a screen limiting the size of the crushed product. Inside the mill is a charge of forged-steel balls, the largest of which may weigh about 70 lb., varying down to the balls of about 2½ in. diameter.

With a suitable charge of ore delivered to the ball space, the rotation of the mill turns the mass of balls and ore over and over and the falling of the balls from shelf to shelf crushes the ore to a size which will issue through the outside screen. While the product from this mill is excellent, objection is found to the shelves and perforated plates as well as outside screens, which require renewal. If we may use the high crushing efficiency of the forged-steel balls and in some way eliminate the perforated plates and screens we may have a very simple intermediate crushing device.

Such a simple and compact crushing device seems possible. The scoop feed in connection with a spiral lining for the feed trunnion of the tube mill is a success. The device known as the tire discharge of a tube mill is a success. This tire discharge permits an opening about 20 in. diameter through which the fine crushed pulps issue. In this 20-in. diameter opening is fitted a reverse spiral so that any pebbles or large particles of ore falling into the grooves are carried back into the mill. Only the particles which have been crushed fine enough to be carried out by the current of water may issue as finished product. If, then, we design a ball mill, using the scoop feed for the feed end and the tire discharge of 20 in. diameter for the discharge end, we are utilizing two well-known and successful elements. A reasonable proportion for the ball mill would be that the diameter be equal to the length of the shell. A mill 4 ft. diameter would be 4 ft. long; a mill 5 ft. diameter would be 5 ft. long.

The liners of the feed and discharge heads may be hard iron or cast steel. The lining for the cylindrical portion of the shell should be of chrome steel or equivalent. The diameter of the balls should be between 3½ in. and 7 in. The larger the feed particle the larger the diameter of the balls.

Ore as coarse as 2 in. diameter may be fed to the ball space of the mill by means of the scoop feed. The mill is to be driven by machine-cut pinion engaging with a machine-cut gear bolted to the feed or discharge head. When the mill is rotated the mass of balls and ore are carried up to about the level of the axis of the shell and the heavy balls rolling down the slope crush the ore. Fine crushed ore issues through the 20-in. diameter tire discharge, any coarse particles or balls worn small being returned to the ball space by means of the reverse spiral inside the tire discharge.

The average results from several types of ball mills show that 1 ton of steel balls will crush about 50 tons of ore during twenty-four hours from a feed of 1½ in. diameter down to a product from 12 to 20 mesh in one operation. Using the same factor, the following table gives the sizes, capacity and power:

Diam. feet.	Length feet.	Wt. of balls in mill lbs.	Wt. of ore in mill lbs.	R.P.M.	H.P.	Capacity tons 24 hrs.
3	3	1035	207	35	7	25
4	4	3360	672	28	25	80
5	5	7650	1530	26	56	190

To compare this proposed ball mill with other types of intermediate crushers, the following table is given, assuming ore to be crushed from 1½-in. feed down to a product passing 12 mesh, No. 19 wire, or about 1 mm.

	Tons capacity 24 hours.	Total wt., lbs.	Price.	H.P.	Wt. of crush- ing mov- ing parts.
6 ft. diam. Chilian...	40-50	50,000	\$3,600	25	20,000
10 stamp mill 850 lbs.	40-50	35,000	2,100	20	8,500
4 ft. x 4 ft. ball mill..	70-80	20,000	1,600	25	3,500

This type of ball mill in connection with crusher and tube mill, making only three crushing units, will reduce ore from run-of-mine to slime pulps passing 150 to 200 mesh; a very simple crushing system for cyanide mills.

The ball mill makes suitable product for table feed, and then the tube mill may recrush the table tailing and middling. The consumption of steel balls amounts to about 1 lb. per ton of ore crushed when crushing dry and 2 lb. per ton of ore crushed when crushing wet.

A review by Mr. S. F. Shaw of nineteen cyanide mills in the United States and Mexico gives the following conclusion:

"Jaw breakers are more favored than gyratory crushers; Dodge breakers are used in only one mill; stamps are almost universal for fine crushing; Bryan, Huntington and Chilian mills are used in a few plants as intermediate grinders following the stamps. Tube mills are being used more and more for fine grinding."

By the use of a ball mill following a Blake crusher run-of-mine ore may be reduced to jig or table pulps in two crushing operations without any screens in the mill system. The small number of mechanical parts; the small mill space occupied; the great range of products which may be produced with simple adjustments; the absence of screens; the crushing of dry ore as well as wet, and the high capacity per unit weight of wearing parts are strong factors for the use of the ball mill in the intermediate crushing operations of our Western mills.

In designs for cyanide mills the diameter of the ball mill and the tube mill might be the same, so that the feed and discharge head castings could be interchangeable between ball and tube mill. The shell lining is bolted to the shell and the large tire discharge opening permits the changing of lining without removing any manholes.

The Gröndal system for the concentration of iron ores uses a ball mill along these general lines, but the feed device is more complicated, and the discharge is through a trunnion bearing. The proposed tire discharge allows a larger and freer discharge as well as a convenient means for renewing the steel liners.

The Hardinge conical mill is a good type for ball mill crushing and has been successful in several installations for intermediate crushing. The Blake crusher and Hardinge conical ball mill followed by the long cylindrical tube mill makes a good combination for cyanide mills in crushing to 150 or 200 mesh.

#### Coarse Concentration by Jigs.

Concentration by jigs includes sizes of feed as coarse as 2½ in. diameter down to sizes as fine as about 12 mesh, No. 19 wire, or about 1 mm. diameter. Formerly finer pulps were concentrated on jigs but since the advent of the reciprocating table, ore pulps finer than 1 mm. are treated on the tables.

The Butte copper ores are jigged as coarse as about 2½ in. diameter, and the rich pieces of ore, including chalcopyrite, chalcocite, enargite and bornite, are subjected to a heavy jiggling action with the result of excessive attrition and liberation of a considerable proportion of slime which causes considerable difficulty in subsequent mill operations. Such coarse jiggling of friable ores might be replaced by improved systems of picking belts and thus prevent the considerable loss through formation of slime by attrition. Power and water are consumed in large quantities in such heavy jiggling operations, and the machinery details need to be very heavy.

When the jig feed is smaller than 1 in. diameter the power and amount of water used are greatly reduced. The average sizes of jig feed in American practice probably lie between 1 in. diameter and 1/32 in. diameter.

Without doubt the jiggling of ores is one of the most interesting and the most complicated of milling problems, combining as it does the study of the crystallization of minerals, specific gravity, cleavage, planes, relative hardness of mineral

and gangue, distribution of mineral contents in the gangue matter, association of minerals of nearly equal specific gravities, and endless combinations which often present baffling problems. Added to these questions are the various jig adjustments referring to stroke, speed, depth of bed, number of jig compartments, relation of screen aperture to feed particle, and combinations varying with each ore, all requiring careful and persistent study.

The familiar type of Harz jig found in our western mills consists of one to four compartments, or more—each compartment being 18 to 20 in. wide and 24 to 40 in. long. This Harz jig, under average conditions will treat about 1½ tons of feed per square foot of jig screen area per day of 24 hours. The large jigs of the Joplin district are often found with seven compartments or cells, each compartment having an area of about 20 sq. ft., or a total of 140 sq. ft. of jig screen area. These seven compartment Joplin jigs treat at the rate of 700 tons per day of 24 hours, or 5 tons of feed per square foot of screen area. The Hancock jig, the feature of which is the movable screen, is a successful type for large capacities and will treat about 10 to 15 tons of feed per square foot of jig screen area daily.

All these jigs employ both a pulsion and suction cycle of water through the jig screen. The average water consumption for Harz jigs in American practice is about 4700 gal. of water for every ton of ore treated. The Hancock jig uses from 350 to 800 gal. of water per ton of ore treated; a very low water consumption.

A few years ago a series of tests on the Richards pulsator jig showed in practical mill operations that 1 sq. ft. of screen area will treat 100 to 180 tons of ore in 24 hours as compared with:

Harz Jig, 1½ tons feed per square foot of screen area.

Joplin Jig, 5 tons feed per square foot of screen area.

Hancock Jig, 10-15 tons feed per square foot of screen area. This high rate of the Richards pulsator jig is made possible by the use of a pulsion stroke only. In the earlier designs the ore bed was 10 to 14 in. deep, requiring about 30 ft. head of hydraulic water at the rate of 1000 to 1200 gal. per ton of ore treated.

The later designs make use of an ore bed of but 4 or 5 in. deep and a consequent reduction of the head of hydraulic water to 10 or 15 ft. This reduction of hydraulic water head has reduced the consumption of water to 600 or 700 gal. per ton of ore treated, or about one-seventh the amount of water required in Harz jigs. The latest types of Richards pulsator jigs also have separate pulsator valves for each compartment, thus giving accurate and wide ranges of adjustment to suit different conditions of ore treatment. Dewatering devices are also placed between the compartments to allow closer and more complete separation between minerals of slight difference in specific gravity. While the theory of this pulsator jig is based on a screen-sized feed, yet in practice a rather wide range of sizes is permitted. The concentrates are particularly clean with a minimum of attrition from wear of concentrate particles in the jig bed. The Richards jigs at the Lewis Mine, Telluride, deliver concentrates ranging from 65 to 70 per cent lead. In the Bonanza mill, Saguache County, lead concentrates are produced as high as 79 per cent lead. In the Cobalt district, Canada, Richards jigs are making concentrates of the smaltite and niccolite ores running as high as 4000 oz. of silver to the ton. A jig installation in Sonora, Mexico, produced argentite concentrates running 400 to 500 oz. of silver per ton. Low grade iron ores of 40 to 45 per cent have been enriched to 65 per cent iron.

There is no doubt about the favorable application of this pulsator principle to a wide range of ores. The numerous mechanical details of the Harz jig are replaced by an extremely simple rotating valve requiring a small amount of power and lasting through a long time of service. In several designs of Harz type, great attention has been given to mechanical elements which shall give a pronounced pulsion stroke up

through the screen and diminish as much as possible the suction stroke down through the screen. This differential motion, so-called, is found to give a greatly increased capacity and also reduces the blinding of the jig screen. A study of the elements of mechanism to produce such differential motions will show complication and the necessity for constant attention to adjustments and renewals for wear. The very simple mechanism of the Richards pulsator valve, consisting of but one moving part and entirely automatic action gives this pronounced pulsion stroke with no suction whatever. The recent design having separate pulsator valves for each jig compartment gives additional automatic features allowing any possible jig adjustment.

#### Pulp Classification for Concentrating Tables.

Considerable progress has been made in the application of important principles in the classification of ore pulps previous to table work, but there is much more to study and to do before we reach the reasonable efficiency. Does it pay to consider all these additional devices designed to produce higher recovery? Assume a mill of 100 tons capacity during 24 hours; the crude ore runs \$10 per ton; the mill makes a recovery of 70 per cent. Suppose by the addition of some device an increased recovery is obtained, say 75 per cent instead of 70 per cent. This means an additional creation of value to the amount of \$18,250 per year. In an ore worth \$5 per ton an increased recovery of 5 per cent means an additional saving of \$9,125 per year.

The preparation of fine pulps previous to table concentration involves two general systems; one based on the screen sizing of pulps, producing grains of nearly uniform diameter, and the other known as the hydraulic classification of pulps in which the mineral grains are small and the grains of gangue comparatively large, or the result of equal falling grains of mineral and gangue in a still body of water or an upward current of water. As mill systems increase in capacity the mechanical difficulties found in maintaining fine screening systems become too great and so the tendency is toward the application of the hydraulic principles of classification.

To Dr. Robert H. Richards is due the great credit for the many practical applications of hydraulic classification; for if one will study the results of his 40 years of investigation it will be found that the most successful hydraulic classifiers of to-day utilize some feature developed by him.

The hindered settling design as well as the Richards-Janney are developments of the earlier design known as the vortex classifier.

The Richards pulsator classifier is successful in the classification of a wide range of ores. Recent designs have reduced the number of parts to adjust, so that now only one water-gate valve requires adjustment, whereas in former designs separate water valves were placed on each of the six compartments. The gates between the compartments also have been removed and the adjustable concentrate-discharge gates have been removed, and in their place are spouts which require but one preliminary adjustment.

The use of the screen in this type of classifier has resulted in the saving of high grade shipping products direct from the classifier, but under some conditions of water supply, difficulties are found from the screen blinding and so recently there has been developed a type of classifier utilizing the many favorable features of the pulsator water current and doing away entirely with the use of the screen. This improved form is known as the launder type of pulsator classifier and this design permits the use of any number of pulp classifications to suit the requirements of the mill system.

An ordinary wood launder is used running above and near the feed end of the concentrating tables. Separate sorting columns are provided for each table, being bolted to the bottom of the wood launder, with a separate pulsator valve for each sorting column so that each table may have its own particular adjustment. The only adjustment to make in this

launder type classifier is to determine the diameter of opening in the wood spigot at the bottom of the sorting column.

A novel device for concentration is permitted, in the use of a pulsator jig compartment, which is bolted to the bottom of the wood launder the same as the classifier units. This launder jig has a separate pulsator valve but no concentrate-discharge gate to adjust. A screen is used in the jig compartment and the concentrate discharges down through the screen into the hutch and out through the opening in a wood plug at the bottom of the hutch. The discharging of the grains of free mineral through such a launder jig relieves the coarse tables from a considerable burden and reduces the amount of middling product delivered from the coarse table.

#### Dewatering Devices.

Great improvements over former methods have been made in the development of systems for dewatering pulps and slimes. Too often pulps are delivered to the concentrating tables in too dilute a condition with a consequent loss.

The dewatering of pulps becomes a very important point in the feed to tube mills, for it has been demonstrated that a feed pulp containing about 40 per cent moisture gives the best efficiency in tube mill crushing. In South African practice the Caldecott diaphragm cone is reported to give satisfaction in large installations where large cones may be employed and the mill feed is fairly constant. However, in small mills with fluctuating loads difficulties in adjustment may be met.

The dewatering of jig and table feed-pulps insures a larger capacity and more complete separation of the various products. Slime table work is particularly sensitive to the consistency of the pulps fed to the tables; for a thickened slime pulp is quite necessary for successful recovery. In districts with low or fluctuating water supply the recovery of water from tailings pulps for use again in the mill system becomes a serious necessity. The Tomboy mill in Telluride makes use of a filter plant for the purpose of reclaiming water from the tailings and furnishes clear wash water for the concentrating tables. The Miami mill in Arizona has a very extensive plant of cement settling tanks returning about 75 per cent of the tailings water for use in the mill system.

A serious fault in many mill systems is the great volume of wash water used to sluice the material through the mill. In many cases the use of too flat launders requires extra wash water, which is carried over the jigs and tables. In such cases, the surplus volume should first be eliminated by means of suitable dewatering devices. Mr. Philip Argall, at Stratton's Independence mill, Victor, Colorado, uses a double spiral classifier of his invention for the purpose of dewatering tailings sluiced from the mill, and uses the reclaimed water for further sluicing operations.

#### Tube Mills.

Great progress is being made in the design as well as the operation of this excellent device for the fine crushing of pulps. The tube mill appears to be the most efficient means for the final reduction to slime previous to agitation in cyanide mills.

The use of the scoop feed has overcome the former troubles from wear of packing joints. This simple and automatic feeder receives general endorsement. The spiral lining in the mills using the trunnion feed insures a positive feed to the pebble space of the mill. The heads are usually lined with hard iron plates, and in the trunnion type mills the lining is swelled out around the feed and discharge openings to divert the falling pebbles away from the heads, thus allowing free entrance and discharge of pulps.

Silex or natural flint linings  $4\frac{1}{2}$  and 6 in. thick are used, as well as the hard cast-iron ribbed linings of the El Oro type and other similar designs. The Brown sectional hard iron and manganese lining give good success in Australian practice. The use of machine cut gears on tube mills was first advocated and used in Colorado and now the practice is quite general.

The design known as the tire and trunnion type offers advantages in a large diameter discharge-opening about 20 in., which allows greater capacity and permits more ready access to the interior of the mill. The tire discharge-opening is bushed with a heavy hard iron reverse spiral, which carries small pebbles or oversize pulp back into the pebble space of the mill, only the slime pulp floating out with the water through the discharge-opening. Provision is made for changing the level of discharge for the regulation of pulp sizes, by varying the diameter of the tire discharge-opening.

Tube mills are in successful operation for the recrushing of jig and table middlings for further treatment on fine and slime concentrators. In the Lewis Mill, Telluride, a 4-ft. diameter by 10-ft. long trunnion type tube mill regounds the coarse table middlings from a 60-ton mill-feed, in 24 hours. At the Bingham Junction, Utah, mill of the United States Smelting, Refining and Mining Company, a 5-ft. diameter by 14-ft. long tube mill, trunnion type, recrushes jig and table middlings at the rate of 136 tons in 16 hours, or 204 tons in 24 hours, delivering a product passing a 40-mesh screen with but 10 per cent of the pulp finer than 200 mesh. This tube mill may be used to produce a granular pulp with but a small percentage of slime for subsequent table work, as well as slime pulps for cyanide practice. The tube mill is a remarkable all-round machine for fine crushing and there may be further developments which will serve well in our mill systems.

Here, then, are a few points of interest in the progress of mill design. The subject is great, and fast multiplying into many branches of marked importance. In spite of what we may desire, our mill systems for the treatment of more complex or lower grade ores, must of necessity require additional devices to care for them; and then a greater intelligence and skill on the part of the operators, and a hearty co-operation between designer, manufacturer, mill superintendent and the man on the mill floor.

The manufacturers always appreciate the helpful suggestions of the men who are in the thick of the mill work, and will do their best to return in good measure improvements in machinery details which will make necessary adjustments easier for the mill man, and renewals and repairs as small as possible.

*Denver Engineering Works Co.,  
Denver, Colo.*

#### The Assay of Ores Containing the Platinum Group of Metals.

BY A. C. DART.

In METALLURGICAL AND CHEMICAL ENGINEERING, February, 1911, page 75, there appeared an article on the concentration of platiniferous copper ore at the Rambler Mine, Wyoming. The article contained a brief statement of methods of assay and analysis used in obtaining the results there given, with due recognition of the fact that the methods were incomplete and that the results might, therefore, be open to some question.

As a matter of fact, it was impossible at that time to make use of commercially accepted methods because they were not available; and it was not until some months later that the writer secured, from various sources, information regarding commercial methods.

The following procedure was finally adopted for the assay of platinum-bearing ore and concentrate produced at the Rambler Mine, and formed the basis of settlement of a number of shipments. For other classes of material certain changes would be necessary in the method of obtaining the first button. Suitable quantities of ore or concentrate are taken, depending on the richness of the product.

Eight portions of one assay ton each are fused with a suitable flux in 20-gram crucibles. If considerable copper is present, as in Rambler ores, the resulting lead buttons are scorified until soft and then cupelled.

The precious metal buttons obtained by cupellation are weighed, giving the total quantity of precious metals in the ore. They are then combined in two check lots, representing four assay tons each, and re-cupelled with ten times their weight of pure silver. The resulting buttons are then parted separately with 12 per cent nitric acid, and finally with concentrated nitric acid in test tubes placed in boiling water. Silver, palladium, and platinum are thus dissolved, and gold remains. The residue is washed by decantation in the usual way, ignited, and weighed, giving the quantity of gold contained in four assay tons of pulp.

The nitric acid solution obtained from the foregoing step is kept as small in bulk as possible. Hydrochloric acid is added to precipitate the silver as chloride, which is filtered and washed. The precipitate may be reserved for a future check.

The filtrate from the silver precipitation is then evaporated to dryness. The residue is moistened with a few drops of hydrochloric acid and taken into solution with 10 cc hot water. The solution is allowed to stand for half an hour, and if silver chloride separates it is again filtered and washed, the precipitate being combined with the first silver chloride obtained.

The filtrate is now made ammoniacal, then acidified with formic acid and boiled. This will precipitate metallic platinum and palladium. Usually no precipitate forms until after boiling begins, and ebullition should be continued for half an hour. The precipitate is filtered, washed, and ignited. The weight gives the combined quantity of platinum and palladium.

After weighing, the platinum and palladium are converted to chlorides by dissolving in aqua regia, evaporating to dryness and taking up with a few drops of hydrochloric acid and 50 cc hot water. This solution is saturated with pure, crystallized ammonium chloride; 10 cc alcohol is added, and the whole allowed to stand for not less than 24 hours. The precipitated ammonium platinic chloride is then filtered, washed with a saturated solution of ammonium chloride, and ignited very slowly on account of the possible volatilization of the precipitate. The resulting metallic platinum is then weighed.

By this method gold and platinum are determined directly. Silver is determined by difference between the weight of the original buttons, and the combined weights of the gold and precipitated platinum and palladium. Palladium is determined by difference between the combined weight of platinum and palladium, and the platinum precipitated by ammonium chloride. A direct determination of silver may be made on the chloride precipitate if desired.

#### Calculation of Furnace Charges—IV.

BY REGIS CHAUVENET.

Some illustrations of the application of stoichiometric principles to problems cognate to smelting were promised, and are here inserted, to serve as types of method rather than of practical cases. They contain, nevertheless, useful demonstrations of the facility with which a small knowledge may be applied. Problems not dissimilar are often presented in practice and may be solved by application of the same elements.

(a) A zinc concentrate containing 90 per cent of zinc sulphide and 10 per cent silicious gangue, is roasted, being converted into a mixture of zinc oxide and zinc sulphate ( $ZnO$  and  $ZnSO_4$ ). It now assays 45 per cent zinc. Find, (1) weight of the product, original concentrate having weighed 100 lb.; (2) weights of the  $ZnO$  and  $ZnSO_4$  in the roasted material.

*Solution.*—Taking atomic weight of zinc as 65, the percentage of zinc in the original material was 60.3. This is 45 per cent of 134, answering first part of the question. For the zinc remains the same in weight, as does the gangue. Weight of product 134 lb.

To ascertain the respective weights of the two compounds, it is easiest to refer to the actual weight of the zinc contained, i.e., 60.3 lb.

Per cent of zinc in  $ZnSO_4$  is 0.4037. Per cent of zinc in  $ZnO$  is 0.8025 (that is, taking unity = 100 per cent).

Let  $x$  = weight in lb. of the  $ZnSO_4$ . Then  $124 - x$  = weight of the  $ZnO$  (for since the gangue weighs 10 lb., the zinc salts weigh 124 lb.).

$$\text{We have then: } 0.4037 x + 0.8025 (124 - x) = 60.3.$$

$$\text{Hence, } x = ZnSO_4 = 98.32 \text{ lb., and } ZnO = 25.68 \text{ lb.}$$

$$\text{Proof. } 25.68 \times 0.8025 = 20.68200$$

$$98.32 \times 0.4037 = 39.691784$$

$$\text{Sum} = 60.299984 \text{ or } 60.3 \text{ lb.}$$

We have seen problems identical as to principle of solution declared to be too troublesome to bother with, by practiced men.

To repeat the substance of a remark made early in this series, chemical relations, easy to understand and apply, having never been properly inculcated, or their elements having "rusted," a singular mystery has been flung over all such problems, slag calculation included, which a little serious study would dispel forever.

(b) The following exercise, somewhat fanciful as to data, is given merely for verification by anyone sufficiently interested in the applications of stoichiometry to work it out. As in (a), albeit this particular problem is of no practical bearing, its solution will aid in working out many simpler questions of quite similar data and method.

An ore has the following analysis: (assume 100 lb.)

$SiO_2$	....	31.48 per cent....
$FeS_2$	.... 30.00	" " Fe ... 14.00 .. S .. 16.00 per cent
$PbS$	.... 19.12	" " Pb ... 16.56 .. S .. 2.56 " "
$ZnS$	.... 19.40	" " Zn ... 13.00 .. S .. 6.40 " "

100.00

Total sulphur = 24.96 " "

This ore is roasted until it contains no more sulphides, losing 10 per cent of its lead and 20 per cent of its zinc in fumes.

All of the iron has now become  $Fe_2O_3$ . One-half of the remaining lead goes to  $PbO$  and one-half to  $PbSO_4$ . One-third of the remaining zinc goes to  $ZnO$  and two-thirds to  $ZnSO_4$ .

What is weight of the roasted product?

What is summation analysis of same?

*Solution.* Open a debit and credit account, setting losses against gains in accordance with the conditions specified. Gains are all in oxygen.

Methods having been given in our "Chemical Introduction," we here simply tabulate the results, distinguishing between oxygen in oxides and in sulphates.

	Losses.	lb.	Gains.	lb.
$FeS_2$ loses $S_2$ .....	16.000	.... O (in $Fe_2O_3$ ) .....	6.00	
$PbS$ loses $1/10$ of $Pb$ ..	1.656	.... O (in $PbO$ ) .....	0.576	
$PbS$ loses $1/10$ of $S$ ....	0.256	.... O (in $PbSO_4$ ) .....	2.304	
$PbS$ loses $1/2$ of $9/10$ of $S$ .....	1.152	....		
$ZnS$ loses $2/10$ of $Zn$ ...	2.600	.... O (in $ZnO$ ) .....	0.853	
$ZnS$ loses $2/10$ of $S$ ....	1.280	.... O (in $ZnSO_4$ ) .....	6.826	
$ZnS$ loses $1/3$ of $8/10$ of $S$ .....	1.706	....		

Total loss .....

Subtract gain .....

Total gain .....

16.559

Net loss .....

8.091

Original weight 100 lb., present weight 91.909 lb.

Composition (lb.) of product:

$SiO_2$ .....	31.480 per cent
Fe .....	14.000 " "
O (in $Fe_2O_3$ ) .....	6.000 " "
Pb .....	14.904 " "
O + O <sub>2</sub> (with Pb) .....	2.880 " "
S in SO <sub>4</sub> .....	1.152 " "
Zn .....	10.400 " "
O + O <sub>2</sub> (with Zn) .....	7.679 " "
S in SO <sub>4</sub> .....	3.414 " "
	91.909 " "

## Analysis of same:

$\text{SiO}_2$	.....	34.251 per cent
Fe	.....	15.232 " "
Pb	.....	16.216 " "
Zn	.....	11.316 " "
S	.....	4.968 " "
O	.....	18.016 " "
	—	
		99.999 " "

## Summation analysis by compounds:

$\text{SiO}_2$	.....	34.251 per cent
$\text{Fe}_2\text{O}_3$	.....	21.760 " "
$\text{PbO}$	.....	8.735 " "
$\text{PbSO}_4$	.....	11.868 " "
$\text{ZnO}$	.....	4.701 " "
$\text{ZnSO}_4$	.....	18.670 " "
	—	
		99.985 " "

(c) An iron ore has 88 per cent  $\text{Fe}_2\text{O}_3$  and 0.8 phosphoric anhydride ( $\text{P}_2\text{O}_5$ ). In smelting it produces, with its fluxing material:

## 1. Pig metal whose analysis shows in part:

Carbon	.....	3.00 per cent
Silicon	.....	2.80 " "

Pig metal also contains all of the phosphorus, reduced from the  $\text{P}_2\text{O}_5$  of the ore (take atomic weight of phosphorus at 31).

2. Slag, whose weight is one-fifth of the weight of the ore charge, and which contains 3 per cent of iron oxide ( $\text{FeO}$ ) derived from the ore.

What weight of the ore produces 100 lb. of the pig metal?

What is the completed analysis of the pig metal?

*Answer.* Weight of ore which produces 100 lb. pig = 153.2 lb.

## Analysis of the pig metal:

Iron	.....	93.66 per cent
Carbon	.....	3.00 " "
Silicon	.....	2.80 " "
Phosphorus	.....	0.54 " "
	—	
		100.00 " "

## Mixing Ores for a Given Percentage.

It is sometimes necessary to mix ores or other material, so that the mixture shall contain a certain percentage of a given substance. Calculations for a furnace charge start, of course, from the analysis of the mixture.

The formulation of this little problem is readily translated into the following:

*Rule.* Take weights of the ores in inverse ratio to the differences between their assays and the (required) assay of the mixture.

*Example.* (a) Mix two ores containing respectively 40 and 8 per cent of a certain constituent, so that the mixture shall contain 16 per cent.

No. 1. The larger per cent. less the required per cent. is  $40 - 16 = 24$ .

No. 2. The required less the smaller per cent. is  $16 - 8 = 8$ .

Now take weights of the two ores inversely as these figures, i.e., take 8 parts of No. 1 and 24 parts of No. 2 (or in this case, reducing to simpler ratio, 1 part of No. 1 and 3 parts of No. 2).

*Proof:*

100 parts of No. 1 contain.....	40 lb. (parts)
300 parts of No. 2 contain (3 $\times$ 8).....	24 lb.

400 parts of the mix contain.....	64 lb.
-----------------------------------	--------

And 64 is 16 per cent. of 400 as required.

If we have three lots, there is no definite solution. The problem is indeterminate, unless there are but two lots to be mixed, since the required percentage must fall either between No. 1 and No. 2 or between No. 2 and No. 3.

We may, however, by limiting weight of one lot to a certain figure, or by stipulating a certain proportion as between cer-

tain classes of ore, reduce the problem to a determinate one. Cases of this kind are exceptional, but it is well to have a ready method of figuring on them.

(b) Three ores assay respectively 39, 24 and 8 per cent of a certain constituent. We want 16 per cent in the mixture. We must use both 1 and 2 and the latter in twice the quantity of the first. Assume for convenience 100 lb. of No. 1.

100 lb. of No. 1 contain .....	39 lb. of substance
200 lb. of No. 2 contain .....	48 lb. of substance

300 lb. of the mix contain.....	87 lb. of substance
---------------------------------	---------------------

The "mix" then of 1 and 2 contains:

$$\frac{87 \times 100}{300} = 29 \text{ per cent}$$

Now combine the "mix" with ore No. 3 as in previous example.

**Method of Representative Equations.**

All methods in common use hitherto have depended upon one or the other of two procedures. In the one, adjustment of matte and bases having been made, if necessary, the pre-established ratio of acid to base is applied by the use of "excess," whether chemical or arithmetical, according to formula asked for. In the other, after the same preparation, formula is ignored, at least in form, and a given ratio of constituents, sometimes confined to that of silica to bases, sometimes of silica plus alumina to bases, often also of bases to one another, is sought.

It has been pointed out that many "type" cases, which are usually stated in percentage form, were in fact originally derived from formulas. In our last article (No. V) we shall touch upon such cases, showing at the same time how a calculation may be made for them without getting the formula into the conventional analytical form.

There are variants in method, in the above named two procedures, into which we do not propose to go. They all depend at last upon some application of the "excess," and it is the main object of this series to show that complex cases are the ones which should drop all use of the excess calculations.

It has already been noted that the problem is easy when there is but one fluxing material to be added, complex when there is more than one. It is often imperative to use two fluxes, and whenever this is necessary it is almost invariably the case that a ratio between bases is needed.

The "bases" range themselves into the "iron" and "lime" groups, as already explained. Let us further explain, that in the type examples, (a) to (e) with which we set forth the general method, we clear ourselves of complications at the outset, by tacitly assuming that matte has been "taken out," also that all adjustments of bases have been made. Thus, we present but two bases, viz.: iron and calcium. Furthermore, these are examples of a method, not of type slags. If a method of computation is correct, it makes absolutely no difference, so far as understanding it is concerned, whether the case is common, or exceptional, or even whether it is probable. If the reader can work one case, in the arithmetical or algebraic sense, he can work as well with any other data.

Notwithstanding the fact that the "Equation" method is more applicable to complex cases, we simplify a first exposition by selecting problems of no complexity whatever.

It must be clearly understood that the method is equally applicable to formulistic or percentage slags, but that the formula, once translated into either ratio or percentage, no longer figures in the calculation.

How to make such transformations has been shown in the "Chemical Introduction," but it is a matter familiar to any chemist or metallurgist.

*Examples.* (a) Given an ore, whose fluxing constituents are as follows:

Silica .....	35.00 per cent
Iron oxide ( $\text{FeO}$ ) .....	20.00 " "

The only fluxing material is a limestone whose analysis is:  
 Silica ..... 10.00 per cent  
 Lime (CaO) ..... 50.00 " "

Take here, as in all the other problems to follow, 100 lb. of ore.

Requirement for the slag, Silica 40 per cent, bases 60 per cent. Call weight of limestone to be used  $100x$ .

Since we use 100 lb. as basis of calculations, any numerical expression will denote either pounds or percentage, reckoning on the ore.

The use of " $100x$ " instead of " $x$ " will presently be seen to simplify the statement.

Since  $100x$  is weight in lb. of the limestone,  $10x$  must be the weight of its silica and  $50x$  the weight of its lime (CaO). That is, if we write analysis of our flux as usual in analytical form, i.e., in "parts in 100" it is evident that by taking as coefficients of  $x$  the percentages in that analysis, we have, if we call the whole weight " $100x$ " correct expressions at once for each constituent. Hence the name we give the method, "representative equations," for the coefficients exactly represent the percentages from the analyses.

Assemble all expressions for silica and for bases, and equate according to the requirement.

	Silica.	Bases.
In ore .....	35 .....	20 .....
In limestone.....	10x .....	50x .....

$$\text{Total silica} = 35 + 10x \quad \text{Total base} = 20 + 50x$$

The requirement is:

$$\text{SiO}_2 : \text{Base} = 40 : 60$$

That is:

$$35 + x : 20 + 50x = 4 : 6$$

Whence,  $x = 0.9286$ .

$100x = 92.86$  = lb. of limestone required.

The "proof" consists in taking 100 lb. ore and 92.86 lb. limestone, and computing their actual weights of silica and base, thus:

Proof.

	SiO <sub>2</sub> .	Base.
In 100 lb. ore.....	35.	20.
In 92.86 lb. "stone".....	9.286	46.43
	44.286	66.43

$$44.286 : 66.43 = 40 : 60$$

No one in practice would "bother" with second and third decimals of a pound, but for the sake of an absolute check, or proof, we carry value of  $x$  to fourth decimal, so that we get value of " $100x$ " to second decimal, and final check comes within one-tenth of a pound.

Let us now do the above little problem by "excess."

(1) Ore. (Since silica is in excess, put base first in the proportion.)

$$6 : 4 = 20 : 13.33 = \text{silica required by its own base.}$$

$$35 - 13.33 = 21.67 = \text{silica excess in the ore.}$$

(2) Limestone:

$$4 : 6 = 10 : 15 = \text{base required by its own silica.}$$

$$50 - 15 = 35 = \text{basic excess of the limestone.}$$

(3) Base required by silica excess of the ore:

$$4 : 6 = 21.67 : 32.505 = \text{base required.}$$

(4) Find weight of limestone required, by adjusting excess in ore to excess in limestone:

$$\frac{32.505 \times 100}{35} = 92.86$$

Proof. As the proof is identical it would be mere waste of space to repeat it.

This case is so elementary that it might be a matter of indifference whether to adopt the "excess" or the "equation." From this example, however, the two methods part company, i.e., it will be found that the equation method solves with fewer and fewer figures, relatively, as the complexity rises.

We now pass to the consideration of one ore and two fluxes, which involves also the two ratios, i.e., silica to bases and bases to one another.

(b)

	Ore.	Iron flux.	Limestone.
SiO <sub>2</sub> .....	30	14	10
FeO .....	10	77	..
CaO .....	..	..	50

This rises from four constituents to be figured, to six. We could insert a case with five constituents, but the illustrations are ample as it is.

The requirement as to composition of the slag is:

SiO <sub>2</sub> .....	40 per cent
FeO .....	40 " "
CaO .....	20 " "

$$100 " "$$

The use of two additional materials requires the use of two equations.

Call weight (i.e., lb.) of the iron-ore flux required  $100y$

Call weight (i.e., lb.) of the limestone required  $100y$

Now "assemble" the constituents.

$$\text{Total silica} = 30 + 14x + 10y$$

$$\text{Total FeO} = 10 + 77x$$

$$\text{Total CaO} = 50y$$

$$(SiO_2 = FeO) \quad 30 + 14x + 10y = 10 + 77x \quad (1)$$

$$(2 \times CaO = FeO) \quad 100y = 10 + 77x \quad (2)$$

Multiply (1) by 10 we get  $553x = 210$ .

$$x = 0.3797$$

$$100x = 37.97$$

$$y = 0.3924$$

$$100y = 39.24$$

$$\begin{aligned} \text{Iron-ore flux required} &= 37.97 \text{ lb.} \\ \text{limestone required} &= 39.24 \text{ lb.} \end{aligned}$$

Proof.

	SiO <sub>2</sub>	FeO	CaO
In ore .....	30	10	....
In iron flux .....	5.32	20.24	....
In limestone .....	3.92	....	19.62

$$39.24 \quad 39.24 \quad 19.62$$

These numbers are in the required ratio, 4 : 4 : 2.

As the number of constituents to be taken into the account rises, the number of terms in the equations must also increase, but the number of unknown quantities remains two.

Problems involving four distinct substances in the slag, i.e., not merely as a physical fact, but as matters for separate adjustment, are not only very difficult to compute, but, what is more to the purpose, are usually impossible. That is, the materials will not lend themselves to the requirement. The method under discussion can be extended to these cases by the use of another unknown and the solution of three simultaneous equations. Usually it will result in a "minus" value, the accepted indication that we are trying to reconcile incompatible conditions. These cases are hardly worth consideration in detail. We pass now to the case of two basic groups in the ore.

(c)

	Ore.	Iron flux.	Limestone.
Silica .....	30	10	5
FeO .....	10	80	..
CaO .....	10	..	53

Requirement, same as in (b), viz.:  $SiO_2 = 40$ ;  $FeO = 40$ ;  $CaO = 20$ .

$$\text{Total silica} = 30 + 10x + 5y$$

$$\text{Total FeO} = 10 + 80x$$

$$\text{Total CaO} = 10 + 53y$$

$$(FeO = 2 \times CaO) \quad 10 + 80x = 2 \times (10 + 53y) \quad (1)$$

$$(SiO_2 = FeO) \quad 30 + 10x + 5y = 10 + 80x \quad (2)$$

These are most readily solved by multiplying (1) by 7 and (2) by 8.

$$(1) \quad 80x = 10 + 106y$$

(2)  $70x = 20 + 5y$  the reduced forms, become:

$$560x = 70 + 742y$$

$$560x = 160 + 40y$$

$$x = 0.2049. \quad 100x = 20.49 = \text{required iron flux}$$

$$y = 0.1282. \quad 100y = 12.82 = \text{required limestone.}$$

*Proof.*

	Silica	FeO	CaO
In ore .....	30.00	10.00	10.00
In iron flux .....	2.949	23.59	.....
In limestone .....	0.641	.....	6.795
	—	—	—
	33.59	33.59	16.795
	40	: 40	: 20

It is seen that the statement and solution are about as quick with the additional base introduced as in (c). The only difference is in the adding in of an additional term involving the unknown, the solution is no more troublesome.

(d) We now introduce two bases into the "iron flux," leaving two in the ore, as in (c).

Ore	Iron flux	Limestone
Silica .....	40	10
FeO .....	20	75
CaO .....	10	6

Let the requirement now be  $\text{SiO}_2 = \text{FeO} = \text{CaO}$ , i.e., each 33½ per cent.

Assemble the expressions for constituents as before: there are more terms, but no complications.

$$\text{SiO}_2 = 40 + 10x + 7y$$

$$\text{FeO} = 20 + 75x$$

$$\text{CaO} = 10 + 6x + 5y$$

$$(\text{SiO}_2 = \text{FeO}) \quad 40 + 10x + 7y = 20 + 75x$$

$$(\text{FeO} = \text{CaO}) \quad 20 + 75x = 10 + 6x + 5y$$

$$x = 0.3831. \quad 100x = 38.31 = \text{required pounds of iron flux.}$$

$$y = 0.7004. \quad 100y = 70.04 = \text{required pounds of limestone.}$$

*Proof.*

$\text{SiO}_2$	FeO	CaO
In ore .....	40.00	20.00
In "flux" .....	3.83	28.73
In limestone ...	4.90	.....
	—	36.42
	48.73	48.73
	48.72	48.72

Weights all equal, as required by the condition.

#### Completed Form of the Problem

(e) We finally make the supposition that the limestone will carry a base of the iron group. This may be a very improbable case, but we have already indicated that these are examples in method, not attempts at physical probabilities. We now have the "complete" problem, that is, each and every one of the materials to be smelted contains silica and two basic groups.

Let us here ask the student or reader to try this problem by "excess." It becomes more like a Chinese puzzle than a problem in arithmetic. The first trials will result in a readjustment, approximating somewhat the desired weights, then, proof being applied, a revision and another approximation. This may be continued until the worker is satisfied that any closer result would be beyond reasonable demands of practice.

Ore	Iron flux	Limestone
$\text{SiO}_2$ .....	40	10
FeO .....	20	72
CaO .....	10	50

*Requirement.*  $\text{SiO}_2 = 40$ ;  $\text{FeO} = \text{CaO} = 20$  per cent.

As in all the problems, let  $100x$  = pounds iron flux,  $100y$  = pounds limestone.

$$\text{SiO}_2 = 40 + 10x + 4y$$

$$\text{FeO} = 20 + 72x + 6y$$

$$\text{CaO} = 10 + 10x + 5y$$

$$(\text{SiO}_2 = \text{FeO}) \quad 40 + 10x + 4y = 20 + 72x + 6y$$

$$(\text{SiO}_2 = 2 \times \text{CaO}) \quad 40 + 10x + 4y = 2 \times (10 + 10x + 5y)$$

We omit all operations and state values at once:

$x = 0.3169. \quad 100x = 31.69 = \text{weight in lb. of required "iron flux"}$

$y = 0.1753. \quad 100y = 17.53 = \text{weight in lb. of required limestone.}$

*Proof.*

	$\text{SiO}_2$	FeO	CaO
In ore .....	40.00	20.00	10.00
In "flux" .....	3.17	22.82	3.17
In "stone" .....	0.70	1.05	8.765
	—	—	—
	43.87	43.87	21.935

These numbers are exactly as  $40 : 40 : 20$ , as required.

The above examples may serve as our "types" for illustration of the method.

The use of " $100x$ ," etc., is recommended for the avoidance of fractional coefficients. The analyses given should always be revised and the percentages raised or lowered to the nearest unit. This for all purposes of slag calculation is amply close. For example, suppose an ore shows, as to slagging constituents:

$\text{SiO}_2$ .....	39.8
FeO .....	17.3
CaO .....	9.7

We should not hesitate to use 40, 17 and 10 as coefficients, knowing how trivial would be the difference in the outcome.

Cases may occur in which the material does not admit the formation of the slag called for.

*Example.*

Ore	Flux	Limestone
$\text{SiO}_2$ .....	20	10
FeO .....	40	80
CaO .....	10	..

*Required.*—Slag of  $\text{SiO}_2$ , 40; FeO, 40, and CaO, 20 per cent.

$$20 + 10x + 4y = 40 + 80x$$

$$40 + 80x = 20 + 10y.$$

Upon attempting to solve these equations it will be found that either  $x$  or  $y$  gives a minus value, and, in fact, in any problem of this type a minus result indicates incompatibility of the material with the requirements. The above data are crude enough, and it would hardly require the "setting up" to show its impossibility, but cases might easily arise in which this would not be so readily perceived.

In "setting up" problems on this method it is not necessary to go through all the detail we indicated in solving the above ("a" to "e").

For example, the original analysis may (as simplified by putting units instead of fractions) be written with the unknown quantities annexed to the percentage figures at once, it being always understood that, when two fluxes are used, the weight of the first is to be  $100x$  and of the second  $100y$ .

*Example.*

Ore	Flux	Stone
$\text{SiO}_2$ .....	$8x$	6y
FeO .....	8	$75x$
CaO .....	5	$3x$

Now the "coefficients" are taken directly from the table of analyses. We have limited our statements in these problems strictly to the "analytical" form for slag requirement. The method itself, however, is subject to no such limitation, as we shall presently show.

It is frequently possible, when a chemical formula is desired, especially (as will usually be the case) if it be simple in structure, to pass directly from the formulistic statement to the equation, not only without the intervention of a calculated analysis, but with simpler coefficients. This point, with a few others relating to certain special cases, will be developed in our closing article.

**The Colorado Scientific Society** recently held two meetings at Telluride and Ouray, Colorado, at which practical talks were given to the mining men of those districts. Mr. Frank E. Shepard spoke at Telluride on the subject of modern mill design. At Ouray Mr. W. P. J. Dinsmoor discussed rock drills and air compressors.

### Meeting of Institute of Metals.

#### The Institute of Metals.

The annual general meeting was held on the 16th and 17th of January, in London. The second day was devoted to the reading and discussion of several very interesting papers.

#### Corrosion.

Mr. ARNOLD PHILIP (Admiralty Chemist) read a paper entitled "A Contribution to the History of Corrosion: the Corrosion of Condenser Tubes by Contact with Electronegative Substances," this constituting the first of a series of contributions to the history of corrosion that Mr. Philip has promised to present to the Institute of Metals.

Mr. Philip's paper was devoted to an examination of the relatively small number of cases experienced by the Royal Navy in which localized corrosion occurred in condenser tubes made of Admiralty composition, namely, copper 70, tin 1, and zinc 29 per cent. The author made the interesting statement—interesting because the causes of corrosion have generally been regarded as so mysterious—that the causes of fully 90 per cent of the cases of corrosion observed in the establishments of the Royal Navy had long been known. In the author's opinion the main problems which remain to be solved concerning localized corrosion are, firstly, the explanation of the causes of rather less than 10 per cent of the cases which are now observed; and, secondly, the devising of a general means of preventing these and all other cases of corrosion superior to the method of protector bars, etc., of zinc, aluminium, steel, or iron, as at present employed.

Mr. Philip included in his paper a schedule of queries as issued by the Admiralty to those of its officials who are likely to be able to throw any light on the subject of the corrosion of condenser tubes. This list of questions is very similar to the Corrosion Enquiry Form recently issued by the Institute of Metals, and if all who are experiencing trouble with their condenser tubes would fill up the latter form (which is modeled to some extent on that of Mr. Philip's, and can be obtained from the Institute of Metals, Caxton House, Westminster, S. W.) there is little doubt that Mr. Philip's "to per cent" of mysterious cases of corrosion would be mysteries no longer.

Mr. Philip's paper concluded with a consideration in detail of five very definite cases of corrosion of tubes used in the condensers of battleships, torpedo boats and electric generating plants of the Royal Navy.

Mr. G. D. Bengough, who opened the discussion, said Mr. Philip's experience led to four or five definite conclusions. He considered that the electrolytic theory sufficiently accounted for nearly all cases of corrosion which had come under his own observation. This point was very important, because that theory did not meet with approval in some quarters. Next, with Admiralty brass there were very few cases calling for other explanation than the ordinary causes of corrosion. Possibly a cheaper metal could be produced capable of resisting corrosion to the same extent as Admiralty brass.

Engineer-Rear-Admiral J. T. Corner said the paper showed clearly that the Admiralty brass tubes, when adequately protected, resisted corrosion; and if it occurred the blame would be on the users.

Mr. T. Vaughan Hughes remarked that in Admiralty tubes the corrosion in all cases seemed to be on the lower side in which the sea water was concentrated on emptying the condenser. The electrolytic theory appeared to offer considerable explanation of the effects of coal and ash dust. Information as to the exact shape of the corroded areas would be useful. Ten years' use would be required to produce the changes in the constituents of naval brass which Professor Carpenter, in his experiments on inversion in copper-zinc alloys, had effected in three months. There was a possibility of hitherto unsuspected changes occurring in the constitution of alloys.

Mr. A. E. Seaton mentioned that in the navy no defective

tubes were encountered, while their composition included a small quantity of tin. He agreed with Mr. Philip on the effects of drainage. Where the iron of the condenser had been employed as a preventive the corrosion of the iron was much more severe than that of the brass.

Mr. Anderson considered that electrical methods would solve the problem of prevention of corrosion; and he was experimenting in that direction. A large proportion of some sixty or seventy cases of which he had had experience during the last three months had been in power stations and the mercantile marine; but an appreciable number of these tubes were of Admiralty brass. The effect of an efficient protective mass could only be got when there was good electrical contact between it and the tubes. He had recently been observing the action of protective masses of special material and found that the current generated per block varied from 0.5 amp. to 1.25 amp.; while the average current in the condenser was about 1 amp. per thousand square feet of surface.

Mr. E. Rhead thought there was a possibility of electrical disturbance arising from the liberated gas from the condenser water coming in contact with the metal. The pitting of propeller blades occurred at points of low pressure, just where liberation of gas from the sea water would occur.

Mr. Philip in his reply, said that 90 per cent of the cases he had experienced were explainable by the electrolytic theory. Another factor was the exposure of the bottoms of the tubes to the action of air and sea water when the condenser was not working. He fully recognized the great importance of the unexplained 10 per cent of cases.

#### Inversion in Copper-Zinc Alloys.

A paper by Professor H. C. H. Carpenter followed on "Further Experiments on the Critical Point 470 deg. C. in Copper-Zinc Alloys."

In this paper it is shown that the so-called beta constituent in copper-zinc alloys is to be regarded below 470 deg. C., as an extremely minute and uniform complex of alpha and gamma particles. Its structural stability is so remarkable that even after six weeks' annealing at 445 deg. C. no appreciable coalescence of these particles has been observed in an alloy of exactly the eutectoid composition.

When, however, a few crystallites either of alpha and gamma are initially present in an otherwise pure eutectoid alloy, then, on annealing at 445 deg. C., this stability is comparatively easily destroyed. These crystallites constitute nuclei for the deposition of alpha and gamma, as the case may be, from the mixture, and eventually in both cases the final result is the same, viz.: coarse alpha, and coarse gamma, which can be seen with the naked eye, which have characteristic colors, viz.: alpha pale gold, and gamma pale bluish-grey, and which have the same etching properties.

The peculiarly persistent structural stability of the pure eutectoid alloy can be explained by supposing that at the inversion temperature on cooling the resolution of beta into alpha plus gamma takes place throughout the entire alloy almost, if not quite, simultaneously. This constitutes a favorable condition for the production of the inversion products in an amorphous form, because the forces operative in crystal production do not have time to come into play.

This view is in harmony with certain peculiarities which the freshly prepared alloy exhibits both in polishing and etching, which disappear when it has passed into a visibly crystalline condition. Further, the structural stability of the eutectoid alloy has been so much diminished by the presence of 4 per cent of aluminium that after two days' annealing at 445 deg. C. it was found to have undergone the typical lamella inversion such as is shown by the iron, ironcarbide eutectoid-pearlite. These facts constitute overwhelming evidence that below 470 deg. C. the beta constituent in copper-zinc alloys consists of the two-phase system, alpha plus gamma.

Mr. O. F. Hudson, opening the discussion, said the paper was

\* reply to the criticisms on the author's earlier communication on the subject; and the microphotographs with which it was illustrated seemed to fully prove his point.

Dr. A. G. Gwyer considered that Dr. Carpenter's hypothesis was supported by almost conclusive evidence. A year ago the eutectoid theory and the polymorphic theory were both possibilities; but the author had now proved the correctness of his views.

Dr. Rosenhain asked what evidence the author had to warrant his departure from the ascertained diagram of Shepherd.

Mr. G. D. Bengough had always been convinced that Dr. Carpenter's explanation was right. He noticed that the change from the alpha state to the beta plus gamma was very rapid; and only the segregation of alpha and gamma into definite crystalline structures was slow. If the action were rapid there should be an abrupt alteration in physical properties about 475 deg. C., and if the action were slow the change in mechanical properties would proceed correspondingly.

Mr. F. Johnson remarked that experiments by Webb on locomotive firebox stays of Muntz metal screwed into the copper plate resulted in difficulties at the temperature of the experiment with diffusion of zinc into the copper plate.

Mr. A. E. Thoine had always regarded spontaneous splitting as arising from a condition of the amorphous phase, and the tendency of the metal to revert to its natural soft state. Information as to the length of time required to split up the alpha in 70 — 30 brass would be useful.

Professor A. K. Huntington commented on the absence of any mechanical tests in support of the author's views, and considered that other points required explanation.

Dr. Carpenter replied that the experiments were being continued for the investigation of points on the practical side; and he hoped to present his results next September.

#### Heating of Alloys in Vacuo.

A paper by Professor T. TURNER, on "The Behavior of Certain Alloys when Heated in Vacuo," was then read and discussed.

It was observed, during the course of some experiments conducted a year ago, that on melting brass in vacuo the whole of the zinc volatilizes, leaving the copper. This separation is quantitative, if the heating is not too prolonged and the temperature not above 1200 deg. C.

The behavior of other copper-zinc alloys when similarly heated was, therefore, investigated. A sample of "poisoned" brass, i. e., brass containing iron, lead, tin, arsenic and other impurities, was heated in a porcelain tube in vacuo at 1200 deg. C., and the residue examined. All the zinc, lead and arsenic, and a little of the tin, volatilized, leaving a residue of copper, iron and most of the tin.

Larger scale experiments are also described, in which 70-30 brass was melted in a crucible, transferred to a large and well-lagged desiccator, and the whole quickly exhausted. The residue in the crucible was cooled and analyzed, and the loss in zinc recorded. As a result of these experiments it is suggested that heating in vacuo might be advantageously applied for the refining of crude copper, brass scrap, etc.

"Hard" zinc may be refined by heating in vacuo to 500 deg. C., i.e., to a scarcely visible red heat. Hard zinc is the residue from galvanizing baths, and contains about 5 per cent of iron. At higher temperatures some iron is carried over with the zinc, but at about 500 deg. C. all the iron remains in the residue. As the temperature necessary is so low, and as there is no loss of zinc by oxidation, there is reason to believe such a process would be commercially successful.

Some preliminary experiments on the equilibrium between zinc vapor, copper and copper alloys are also described.

Zinc itself volatilizes easily at 375 deg. C. in vacuo. A 60-40 brass readily loses zinc at 520 deg. C. and a 70-30 brass at 550 deg. C. On heating zinc with copper in vacuo, the metals not being in contact, the copper rapidly turns yellow at 380 deg. C.,

while the zinc is still solid, enough zinc volatilizing to form a surface layer of brass. Zinc distils readily in glass vessels in vacuo, the vapor being colorless and transparent. The zinc condenses in globules, having the appearance of mercury.

Sir Gerard Muntz said it was remarkable that the zinc had entirely separated from the copper; and microscopic investigation might furnish useful data bearing on the mechanical strength of metals.

Mr. T. Vaughan Hughes said that the volatility of metals in a high vacuum came prominently under his notice in connection with work on early incandescent lamps. Even platinum would volatilize in a vacuum; and it was not possible to expose metals to such conditions, even at low temperatures, without volatilization occurring. Zinc could easily be evaporated out of brass without using a vacuum; but it has been discovered that a certain combination of gases prevented that action to a very great extent, and he had been working on these lines.

#### Properties of Alloys at High Temperatures.

Mr. G. D. BENGOUGH presented a paper on "A Study of the Properties of Alloys at High Temperatures."

A series of tensile tests on selected metals and alloys are described. The tests were carried out at temperatures varying from the ordinary temperature to the neighborhood of the melting points of the materials used. A special form of apparatus was used for the tests, and a full description of it is given in the paper.

A number of unexpected results have been obtained. The curves showing the variation of mechanical properties with temperature show certain "mechanical critical points" in the neighborhood of which the direction of the curves alter rapidly, and this phenomenon occurs even in the case of commercially pure metals such as copper and aluminium. A theoretical explanation of the shape of the curves is proposed and discussed.

The paper is illustrated by a large number of curves, tables, and photomicrographs.

Sir Gerard Muntz led the discussion and said that Mr. Bengough seemed to misapprehend the real nature of Muntz metal. It was accepted that anything containing less than 60 per cent copper was not Muntz metal. With class B Muntz metal the author's conclusions were identical with those arrived at by works experience as to the great difference in elongation and ductility as compared with true Muntz metal, which indicated the presence of too much zinc in the metal. His own experience induced a belief that rolled rod would have a longer life and give better results than extruded rod, although at first the latter might give equally good results on mechanical tests; but later a change took place in the crystalline structure and made the extruded rod less reliable. The paper helped to explain this change.

Professor T. Turner thought that the paper suggested a reason why the tensile strength of cast metal fell gradually as the temperature increased; and helped to elucidate problems connected with hardness caused by working. He was of opinion that the amorphous inter-crystalline structure originated in the action of the working between the crystalline areas. As a rule, it was impossible to increase the tensile strength of the material in the cast state to more than double.

#### The Influence of Oxygen on Arsenical Copper.

Mr. R. H. GREAVES presented a paper on "The Influence of Oxygen on Copper Containing Arsenic or Antimony."

The experiments described were made to determine the influence of oxygen on certain mechanical and physical properties of copper containing either arsenic or antimony in quantities up to 0.5 per cent. A number of comparative tests in which the metals were cold-rolled down to 0.02 in., or drawn into fine wire, showed that with increasing arsenic the metal may take up more and more oxygen without suffering deterioration in its capacity for rolling. This quantity of oxygen increases from about 0.05 to 0.2 per cent as the arsenic in-

creases from 0 to 0.2 per cent., then more slowly to about 0.28 per cent, with 0.5 per cent arsenic. Above this point the malleability falls off, and with still more oxygen the metal becomes "cold-short." The action of oxygen on copper containing antimony is similar, but in this case a small quantity (about 0.1 per cent) effects a distinct improvement in the malleability of the metal. The ductility was similarly affected by oxygen. Tensile tests on the case metals showed that increase in oxygen from 0.15 to 0.4 per cent caused a rapid diminution in elongation; the effect on the maximum stress was not marked.

Oxygen has little effect on the hardness (measured by the Shore scleroscope) until a limit is passed, depending on the arsenic or antimony present; above this, the hardness increases rapidly. This limit is at about 0.3 per cent with 0.4 per cent arsenic, but is less in presence of antimony. The first addition of arsenic or antimony decreases the hardness of copper containing oxygen. Measurements of electrical resistance show that oxygen diminishes the conductivity of copper containing arsenic, but increases that of copper containing antimony; the quantitative effect of the oxygen has not yet been determined.

In an appendix, methods of determining oxygen in copper are discussed. Without denying the accuracy of other standard processes, the author suggests that oxygen may conveniently, and with a high degree of accuracy, be determined by reducing in hydrogen, and weighing the water formed.

In the discussion Sir Gerard Muntz said that unless the proportion of oxygen were taken into account, consideration of the effect of arsenic would be useless. His own experience was confirmed by the author's experimental results; but he expected Mr. Greaves' figures for oxygen would be with the decimal point moved one place to the left. When arsenic was 0.5 he should expect the oxygen to be 0.02 to 0.015. With higher oxygen there was a tendency to increase of tensile strength, and rapid reduction of elongation; while with less than 0.015 per cent of oxygen a very ductile soft metal of little practical value was obtained.

Mr. L. Archbutt said he did not consider that any increase of accuracy resulted from the author's method of weighing the water, and it certainly involved much more work and trouble. If railway companies specified the amount of oxygen, there would be great difficulty in working to such a specification. If the arsenic content were specified the oxygen was left to the manufacturer, and when mechanical tests were fully specified other tests were not specified.

Mr. F. Johnson thought that the author had exercised a wise choice in taking arsenic and antimony for experiment. The results agreed with his own previous conclusions that both for hot and cold rolling oxygen greatly improved copper containing antimony. It had been found possible to draw into wire a copper containing more than 3 per cent of antimony and 0.04 per cent of oxygen. Tensile tests of cast copper were unsatisfactory; the wrought material yielded more reliable results. The author stated that oxygen in arsenical copper lowered the conductivity but raised it in antimonial copper. That appeared to confirm the contention that in arsenic-oxygen-copper alloys the oxygen occurred in free CuO<sub>2</sub> and the arsenic was in solid solution.

In the antimonial alloy part of the antimony united with the oxygen instead of going into solid solution.

#### Nomenclature of Alloys.

Dr. W. ROSENHAIN presented a "note on the nomenclature of alloys."

In this paper Dr. Rosenhain raises the question of the nomenclature of non-ferrous alloys and puts forward some tentative suggestions to serve as a basis for discussion. He draws attention to the confusion in nomenclature which exists at the present time, particularly as regards such terms as "brass" and "bronze" which are frequently applied in contradictory ways. A system of nomenclature is put forward for consideration in which alloys are classified according to the system of binary

alloys to which they approximate most closely and class names for such binary systems are advocated—such as the term "brass" for alloys consisting principally of copper and zinc; "bronze" for alloys consisting principally of copper and tin; and so on. The additional metals present in any particular alloy would be indicated by additions to the class name, so that an alloy consisting of copper and zinc with a comparatively small percentage of manganese would be called a "manganese brass." Some of the difficulties which have to be overcome by such a system are pointed out, and the suggestion is made that the whole subject should be dealt with by a representative committee of members of the Institute who should be asked to draw up a system of nomenclature for use in scientific and industrial work, particularly in specifications. Dr. Rosenhain adds that he does not wish to interfere with ordinary trade names used in connection with the advertisement and sale of alloys of special properties, particularly when these are non-descriptive, but generic names such as "brass" and "bronze" should be used accurately, and a satisfactory and widely recognized system of nomenclature is none the less desirable.

#### Tin and Lead in Brass.

Mr. F. JOHNSON presented a paper on "The Effect of Tin and Lead on the Microstructure of Brass."

In this paper the author recorded the results of experiments made with the object of ascertaining the structural relations which exist between lead and tin when present in brass where the ratio of copper to zinc is 2 : 1.

He found that, by itself, tin was thrown out of solution in the alpha phase (70 copper, 30 zinc) in the form of a hard pale-blue compound (probably SnCu<sub>2</sub>) some time after solidification of a slowly-cooled alloy. Annealing, either preceded by rolling or not enabled this compound to again pass into solution in the alpha phase.

In a brass consisting of the alpha and beta phases, where the latter was present in sufficient amount (e.g., naval brass and Muntz metal) the copper-tin compound was much more soluble, being apparently retained by the residual beta.

In an alloy where the beta phase exists in an unstable form (such as the alloy containing 2 parts copper to 1 part zinc) it is insufficient in amount to retain the tin compound which is deposited mainly as a thin envelope or film separating the beta from the alpha phase, thus giving a reticulated appearance to a polished and etched section.

With regard to lead, the author found that this acted independently of the tin exhibiting behavior similar to that of lead in ordinary brass or in copper. That is to say it retained its individuality and existed in small particles unalloyed with more than mere traces of any other metal, these particles tending to segregate toward the areas last to solidify. In the presence of excess of the beta phase lead is probably held in solution therein.

As a result of the observation the author strongly advocates a very thorough annealing of all cast material of the 70-20-1 and 62-37-1 compositions (Admiralty and naval brass respectively) before subjecting it to rolling or drawing, in order that the brittle tin compound may be enabled to pass into solution.

**Malaysian Tin Exports.**—The following table (from the Federated Malay States Government Gazette) shows the weight of tin (tin exported in the form of ore having been taken at 70 per cent of the gross weight of the ore) exported from the Federated Malay States during 1910 and 1911 (picul = 133 1/3 pounds):

States	1910			1911		
	Tin Piculs.	Tin Piculs.	Total Piculs.	Tin Piculs.	Tin Piculs.	Total Piculs.
Perak .....	109,867	311,468	421,335	97,835	339,504	437,339
Selangor .....	43,397	196,795	240,192	54,215	176,960	231,275
Negri Sembilan ..	121	34,576	34,697	90	29,140	29,230
Pahang .....	12,930	27,744	40,674	15,281	28,673	43,954
Total .....	166,315	570,583	736,898	167,421	574,277	741,698

## The Natusius Electric Steel Furnace.

By PROF. W. LIPIN.

Among the electric arc furnaces for steel refining the Héroult furnace has so far found most extended use in practice. There are, however, in operation also quite a number of other types of electric arc furnaces for steel refining, like those of Girod, Stassano, and Keller.

More recently a new electric steel furnace has been designed by Natusius. In practical use in the steel works at Friedenshütte, Upper Silesia, Germany, it has proven very satisfactory and economical. This furnace will be described in the following article. The patents for this furnace are owned by

the Westdeutsche Thomasphosphat Werke, G.m.b.H., Berlin, W. 35.

The first experiments with a Natusius furnace of 1-ton capacity were

FIG. 1.—PRINCIPLE OF NATHUSIUS FURNACE.

made some years ago at the steel works in Friedenshütte and the results then obtained were so satisfactory that in June, 1909, a new furnace of 3 tons capacity was started in operation at that plant. This was soon afterward changed into one of 5½ tons capacity.

The Natusius type is a combination of the Héroult type and Girod type, but embodies also a third novel feature, that is, the heating of the bottom by the current from the bottom electrodes. The Natusius furnace is, therefore, a combined arc and resistance furnace.

Fig. 1 shows the principle of the furnace in its simpler form. The horizontal section through the hearth is circular. From the top are suspended three carbon electrodes, the lower terminals being near the slag layer, as in the Héroult furnace. Three steel electrodes are embedded in the bottom of the hearth. They do not pass, however, all the way through the bottom up to the metallic bath, but only to about the center of the height of the bottom, so that their upper end surfaces are covered with a layer of dolomite of 30 to 35 cm thickness.

The high-tension, three-phase currents which are generated either in a special three-phase generator or supplied from the general power plant of the steel works are fed to the high-tension circuit of a step-down transformer, from the low-tension side of which the electrical energy is supplied to the furnace.

There are three paths of the currents within the furnace: (1) From the top electrodes to the neutral point within the bath through arcs playing between the lower ends of the top electrodes and the bath; (2) from the bottom electrodes to the neutral point within the bath, and (3) from the top electrodes directly to the bottom electrodes. It is very probable that currents also pass directly from one electrode to another in the upper group of electrodes as well as in the lower one.

The arcs playing from the lower ends of the upper electrodes produce a very high temperature in the slag (just as in the Héroult furnace) and cause a very energetic reaction. The passage of the current in various directions causes a very thorough heating of the metallic bath and of the slag, as well as of the furnace bottom. Further, there is a rotation of the metallic bath around the axle of the furnace and a movement of the bath between the electrodes toward the center. These movements are probably due to the use of three-phase currents producing a rotary magnetic field. This is true for the top electrodes as well as for the bottom electrodes. This automatic circulation and mixing of the bath results in a uniform temperature so that the steel made in the furnace is very homogeneous.

The displacement of the neutral point from the transformer or generator into the furnace\* also has the advantage that the arrangement acts like a buffer, in so far as sudden current rushes which may have such a deleterious effect on the generator, are greatly reduced in intensity.

These properties of the Natusius furnace I consider to be important advantages over other arc furnace types.

At Friedenshütte the 6000-volt, three-phase currents of the supply network are transformed by a three-phase transformer down to 110 volts. For connection of the electrodes with the transformer flat copper bars are used, so dimensioned as to carry continuously a current of 2500 amp.

Fig. 2 is a wiring diagram, the figures of the volts and amperes being mean values. *A* represents ammeters, *V* voltmeters and *W* wattmeters. During the normal operation the voltages have the following normal values. Between any two top electrodes the tension is 110 volts, between the top electrodes and the bottom electrodes it is 63 volts, and the tension between any two bottom electrodes is 15 volts.

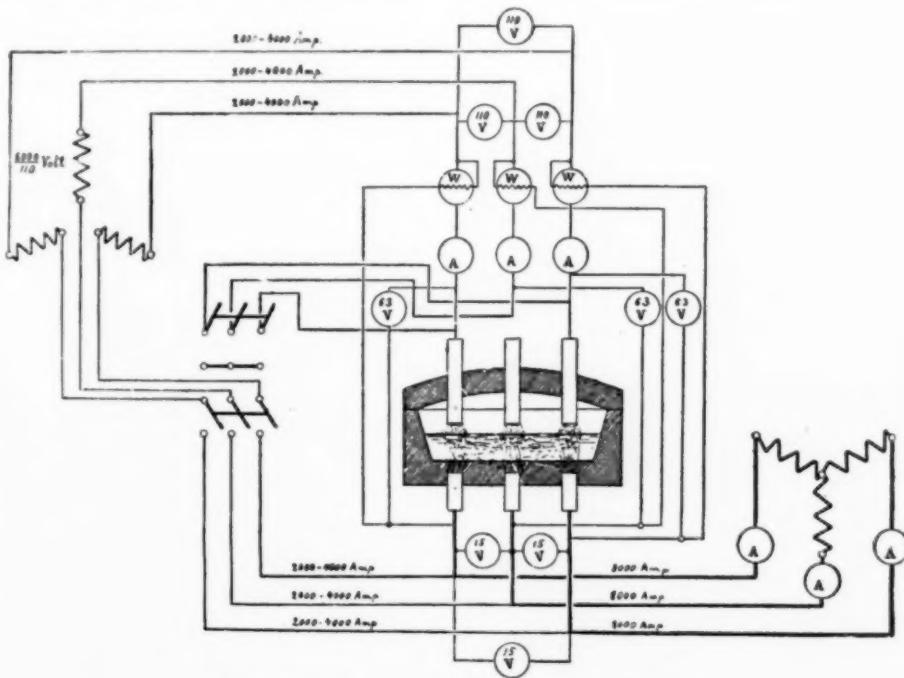


FIG. 2.—WIRING DIAGRAM OF NATHUSIUS FURNACE.

An additional transformer is also provided by means of which the energy supply to the bottom electrodes can be raised at will in order to heat the hearth and bath more strongly from the bottom. However, the absolute necessity of this additional transformer is doubtful. At present the furnace operates very satisfactorily without an additional transformer, since there

\*This is shown in Fig. 2, and further explained in the American patent of Natusius (this journal, vol. IX., p. 608).—Editor.

is automatic regulation of the energy supply to the upper or lower electrodes. This problem is now being studied by the Bergmann Elektricitäts Werke, A. G., in Berlin, and will be solved shortly.

#### Furnace Construction.

The Natusius furnace in operation in Friedenshütte is designed for a capacity of 5 to 6 tons (in the average  $5\frac{1}{2}$  tons). The horizontal cross-section is circular. There are three doors, arranged symmetrically (that is, spaced at angles of 120 deg. with respect to each other).

The door on the front side is provided with a launder for

a circular cross-section with a diameter of 250 mm. They are embedded in the bottom and are also water-cooled.

The whole furnace is enclosed in sheet iron, reinforced by a strong steel ring with two journals. The two journals which are used for tilting the furnace in order to discharge the steel rest in two bearings of a stator frame. Such a tilting arrangement is very convenient, but is suitable only for small furnaces. For larger furnaces with a capacity of more than 6 tons Natusius uses a rack-and-pinion arrangement for tilting. Compressed air is used as the power for tilting at Friedenshütte.

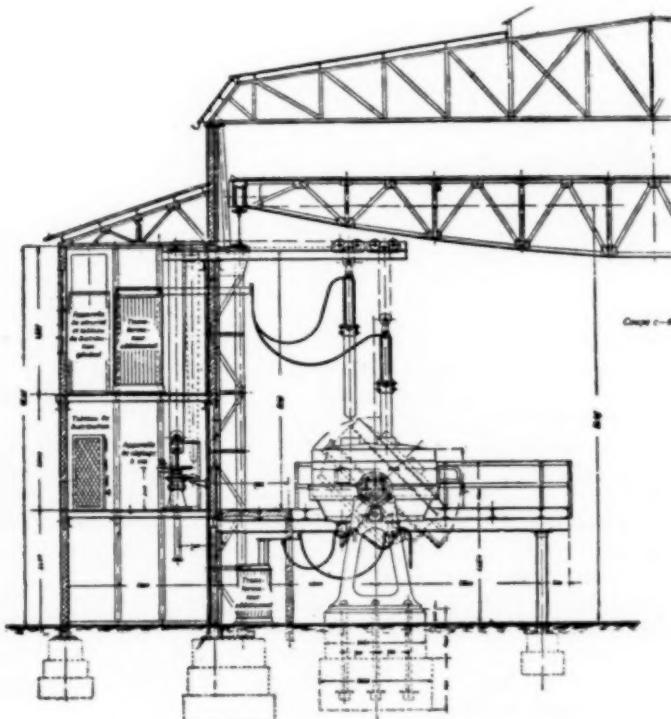
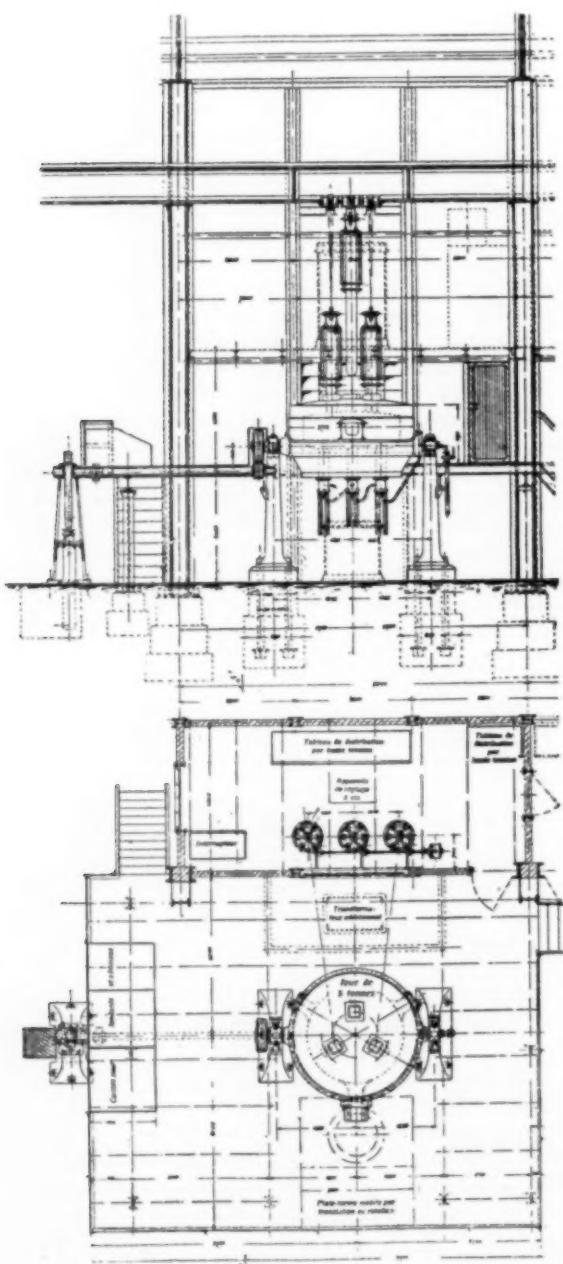


FIG. 3.—SECTIONS OF  $5\frac{1}{2}$ -TON NATHUSIUS FURNACE.

The whole furnace frame, made of sheet iron, is closed. The roof can be easily removed as in the Héroult furnace.

The furnace roof is lined with dinas brick (silica brick) of the Lütgen or Stella (Willisch) brands. The thickness of the roof lining is 250 mm. The thickness of the bottom must not be less than 600 mm. The steel electrodes should extend upwards to about the center of the height of the bottom so that they are embedded in the lower half of the furnace bottom.

The position of the upper carbon electrodes can be regulated by means of cables running over rolls. Current is supplied to each of the upper electrodes through twelve flexible copper laminas and to each of the bottom electrodes through six cables from the main transformer and six cables from the additional transformer.

The upper electrodes are separately water-cooled, i.e., each water jacket has its individual pipes for the supply and exit of the cooling water. The bottom electrodes have a common cooling device, since their water jackets are connected together by pipes.

All measuring instruments are placed in a special room separated from the furnace by a glass wall. The regulation of the distance of the carbon electrodes from the bath is done by hand; the man attending to this has to watch constantly the readings of the voltmeters.

The design of the furnace in operation at Friedenshütte is given in three sections in Fig. 3.

The design of a 12-ton Natusius furnace is shown in Fig. 4. For tilting a rack-and-pinion arrangement is used, the power being supplied by an electric motor. The three upper

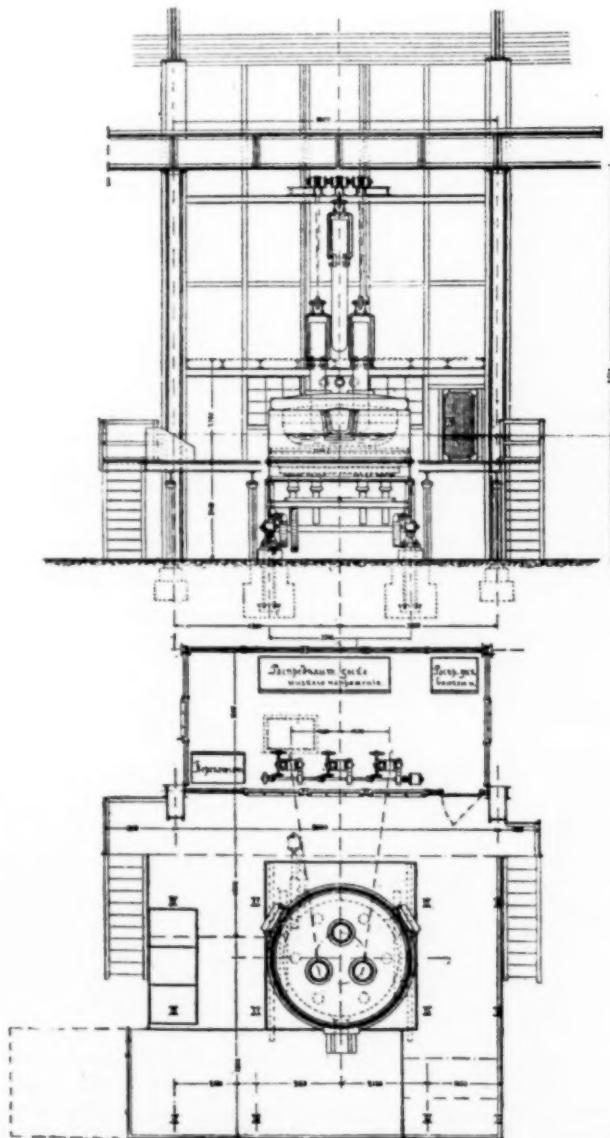
charging molten metal from the basic converter and for discharging the finished steel. Three carbon electrodes are suspended from the top through the arched roof.

The inside diameter of the hearth space is 2170 mm, the height of the center of the arched roof over the bottom of the hearth is 670 mm.

The upper carbon electrodes, which have a cross-section of  $250 \times 250$  mm, are water-cooled. The lower electrodes are cast from steel (without any mechanical machining) and have

electrodes have a diameter of 400 mm. Instead of three bottom electrodes there are provided six steel electrodes, connected in pairs. These ensure a better distribution of the energy in the bath, which is desirable for maintaining a uniform temperature in the bath, especially when the horizontal section is of large dimensions. Otherwise the equipment of the 12-ton furnace is the same as that of the 6-ton furnace.

Concerning the position of the neutral point in the Nathusius furnace and the energy consumed in heating the bottom from the bottom electrodes the Bergmann Elektricitäts Werke gives the following interesting notes:



The most characteristic feature of the Nathusius type of furnace is the possibility of distributing at will the heat effect between the upper and lower groups of electrodes. When a new furnace is to be designed, it may be specified that 50 per cent or more of the total heating effect is to be supplied through the bottom electrodes, and the design of the transformer and additional transformer can then be made accordingly. But even with the furnaces now in existence the heating from the bottom represents quite a considerable percentage of the total heating effect.

Repeated measurements have shown that with 12 volts between the bottom electrodes the power introduced through the bottom electrodes is 32 kw to 38 kw (while the heating effect of the upper electrodes is 110 kw to 150 kw). When the

connections are changed to 24.3 volts the power introduced through the bottom electrodes rises to 90-95 kw (while the heating effect of the upper electrodes is 110 kw to 115 kw).

All measurements show that when a new run begins, that is, when the bottom is relatively cool, the power consumption of the bottom electrodes is less than at the end of a run, although the voltage between the bottom electrodes is a maximum at the beginning. This fact proves that the resistance of the molten bath and of the furnace bottom material decreases more rapidly than the voltage.

On the other hand, it is thereby also proven that the bottom

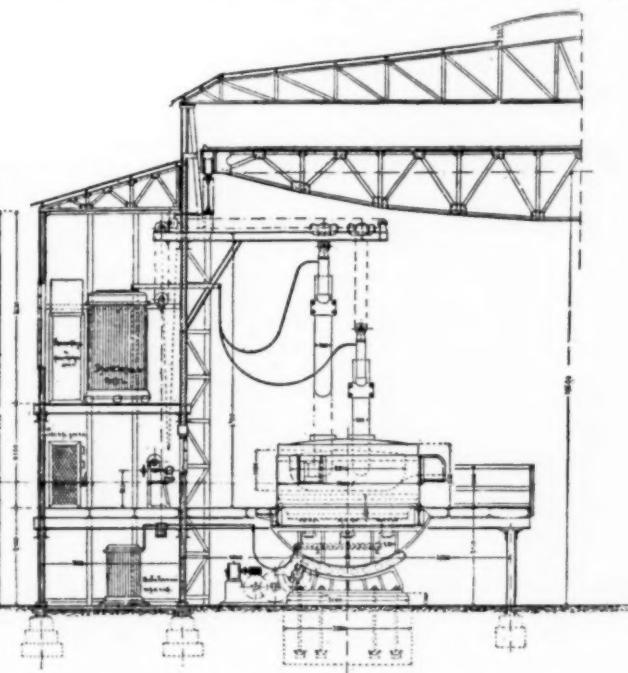


FIG. 4.—SECTIONS OF 12-TON NATHUSIUS FURNACE

material never attains such a good conductivity that it could be considered as a direct short-circuit connection between the bottom electrodes. The neutral point of the bottom electrodes as well as of the upper electrodes is always located in the molten bath.

A direct proof of this was obtained by a test made on Dec. 1, 1910, with electrodes, the upper ends of which were bare. The rammed bottom material above the upper surfaces of the bottom electrodes had been removed so that the three bottom electrodes were electrically connected together through the molten bath. It was found in this case that the voltage between the bottom electrodes did not decrease below 6.2 volts, while the energy consumption was almost the same as when the bottom material had not been removed from the top of the steel electrodes. The measured voltage includes the voltage drop in the leads and in the bottom electrodes themselves. When the furnace was emptied it was ascertained that the upper surfaces of the steel electrodes had really remained bare and had not been covered with any bottom material.

In any case this proves that the neutral point is in the bath and the heating effect of the bottom electrodes represents quite a considerable percentage of the total heating effect.

#### Operation.

The lining of the bottom consists of ordinary dolomite\*, as in the open-hearth. After the furnace has been preheated by a wood or coke fire, a small current is turned on, whereby the bottom is well heated. After the furnace has been properly heated up, the charge is made up.

\*In the American patent of Nathusius (this journal, vol. IX., p. 608) the lining is said to consist of ordinary burned dolomite with tar as binding agent.—Editor.

First 75 kg to 80 kg of iron ore (Kriwoi Rog) and 40 kg to 50 kg of lime are fed into the furnace, whereupon molten metal is charged either from the basic converter or open-hearth as the case may be. After a few minutes the upper electrodes are lowered and the current is switched on.

The electrodes must be lowered gradually, because the pouring of the molten metal on the ore causes a rather energetic reaction and the boiling or spattering of the charge is liable to damage the electrodes.

After the current has been switched on and the boiling has stopped, 50 kg to 70 kg of ore and 70 kg to 80 kg of lime are added to the furnace charge. After about an hour the first sample is taken and if the metal is found to have become sufficiently soft, i.e., if the greatest part of manganese, carbon, and phosphorus has been removed, the furnace is tilted and a part of the slag is run off.

The process is continued in the same way until the samples show the metal to be quite soft; a few shovels of lime are then charged in order to make the slag thicker, the current is cut off, the electrodes are raised, and after the furnace has been tilted somewhat, the slag is carefully removed. The furnace is then tilted back, the electrodes are lowered and the current is turned on again.

To produce a desulphurized slag a mixture of 15 kg of fluorspar, 15 kg of sand, and 40 kg of lime are added. After about 10 minutes a certain quantity of 75 per cent ferrosilicon is added for deoxidation of the metal. The whole desulphurizing process lasts 35 to 40 minutes and during this period the temperature of the furnace is raised as much as possible. The temperature in the bath is tested during this period with the aid of a thin iron rod. When the furnace is sufficiently hot, ferromanganese and special alloys are added and the steel is then poured from the furnace into the ladle. Aluminium is added in the ladle.

With respect to current and voltage, the current per phase during the first period of the process is from 3500 amp to 4000 amp and the power about 200 kw. If the furnace has cooled off somewhat before a new run, the bottom is reheated by means of the auxiliary transformer, the current per phase being 6000 amp to 8000 amp, which for a tension of 15 volts to 20 volts between the bottom electrodes corresponds to 120 kw to 160 kw. At the end of the first period (when the sample is taken), the secondary current of the main transformer is reduced to 2000 amp per phase.

The length of the whole run is from 2½ to 3½ hours. But, as will be seen later on, this is extraordinarily long, on account of the unsuitable location of the plant; the furnace itself, which operates in excellent shape, is not to be blamed on this account.

The energy consumption varies between 270 kw-hours and 290 kw-hours per ton for runs in which ordinary or special steels (for springs, tools, etc.) are made and increases to 400 kw-hours in the production of very soft steel.

Data on a few runs follow.

#### *Run 616. (Special steel.)*

10:25 a. m.—Ore and lime are charged and the molten metal is run into the furnace.

10:40—Removal of part of the slag.

12—Removal of slag.

12:10—Addition of desulphurizing slag.

12:55—Final sample taken. Addition of deoxidants.

1:10—Discharge of finished steel.

The run, therefore, lasted 2 hours and 45 minutes.

In this case the molten converter steel was charged into the electric furnace after it had cooled off somewhat because it had been empty for 50 minutes.

Five thousand and six hundred kg of ingots were obtained; the energy consumption was 1620 kw-hours, or 290 kw-hours per ton of steel.

#### *Run 617. (Special steel.)*

2:10 p. m.—Charge of ore and lime. Charge of molten metal.

3:35—Removal of part of slag.

4:16—Removal of slag.

4:20—Preparation of desulphurizing slag.

5:00—Discharge of finished steel.

The run, therefore, lasted 2 hours and 50 minutes. When the furnace was charged it had been empty for an hour.

Five thousand three hundred and fifty kg of ingots were obtained. The energy consumption was 1140 kw-hours, or 270 kw-hours per ton of steel.

#### *Run 622. (Special steel.)*

11:40—Charge of ore, lime and molten converter metal.

1:45—Removal of part of slag.

2:00—Removal of slag.

2:12—Preparation of desulphurizing slag.

2:50—Addition of ferromanganese and discharge of finished steel.

The run lasted, therefore, 3 hours and 10 minutes. The furnace had been empty for 1 hour and 5 minutes when it was charged.

The specific energy consumption was 290 kw-hours per ton.

In runs making special steels the composition of materials is as follows:

Krivoi Rog ore, 120 kg to 140 kg; lime, 160 kg; fluorspar, 15 kg; sand, 15 kg; ferromanganese, 36 kg.

#### *Run making soft steel.*

12:15—Charge of ore and lime and molten metal.

2:15—Removal of part of slag.

3:30—Removal of slag.

5:10—Addition of deoxidants.

5:10—Discharge of finished steel.

The run lasted, therefore, 4 hours and 55 minutes. The energy consumption was 2400 kw-hours; 6000 kg of ingots were produced. The specific energy consumption was, therefore, 400 kw-hours per ton of steel.

The consumption of materials was as follows:

Manganese ore, 100 kg; lime, 105 kg; fluorspar, 12 kg; sand, 12 kg; ferromanganese, 25 kg; petroleum coke, 18 kg.

In order to show the results of the refining process, I give some analysis of the original converter metal and the finished electric steel obtained in those runs during which I was present in Table I.

TABLE I.

Number of run	Converter Metal				Electric Steel			
	C	Mn	P	S	C	Mn	P	S
616	0.064	0.59	0.065	0.04	0.051	0.14	0.007	0.007
617	0.063	0.27	0.045	0.02	0.057	0.12	0.003	0.015
622	0.066	0.49	0.065	0.04	0.050	0.11	0.002	0.015

Besides these three runs, which I followed very carefully, I give some analyses from the works records in Table II.

TABLE II.

	Converter Metal				Electric Steel			
	C	Mn	P	S	C	Mn	P	S
0.06	0.45	0.06	0.04	0.058	0.12	0.002	0.008	
0.06	0.35	0.03	0.028	0.065	0.12	0.002	0.009	
0.06	0.40	0.05	0.02	0.056	0.085	0.003	0.008	
0.07	0.43	0.06	0.03	0.064	0.14	0.018	0.009	
0.056	0.41	0.028	0.03	0.060	0.19	0.004	0.008	
0.053	0.37	0.04	0.035	0.060	0.12	0.007	0.008	

For soft charges the results of the electric process are also excellent, as will be seen from Table III.

TABLE III.

	Converter Metal				Electric Steel					
	C	Si	Mn	P	S	C	Si	Mn	P	S
0.056	?	0.52	0.04	0.02	0.06	0.01	0.375	0.002	0.008	
0.06	?	0.38	0.035	0.03	0.11	0.15	0.55	0.003	0.008	

The analyses show that the removal of phosphorus and sulphur is accomplished successfully and that the steel obtained is distinguished by its purity.

Of course, the quality of electric steel depends not only on the chemical purity of the metal, but also on the freedom from all blow-holes, etc. It is to be said that in this respect the

electric steel produced in the Natusius furnace is absolutely perfect and is dense and homogeneous even at the ends of the ingots.

#### Chemistry of Refining Process.

In now discussing the chemistry of the refining process it should first be pointed out that in Friedenshütte the time of the refining process is lengthened by delays in the charging of the converter metal and also by the fact that the electric furnace is charged with finished converter steel, which is ready for casting, containing 0.4 to 0.5 per cent manganese. In the electric furnace the whole refining process is lengthened by the time required for the removal of the manganese and the process would undoubtedly work much more rapidly and much more economically if the furnace was charged with metal from a basic converter before spiegeliron or ferromanganese is added to it.

The removal of the sulphur in the electric furnace is as complete as in the Heroult furnace, as will be seen from the analyses.

Dr. Geilenkirchen and Professor Eichhoff contribute the successful removal of sulphur in the Heroult furnace to the formation of calcium carbide, due to the action of the arc on the lime slag, when it contains no longer any metallic oxides, that is, iron and manganese oxides. During the desulphurizing period of the run they believe that the petroleum coke deoxidizes the lime and converts it into carbide according to the equation



The calcium carbide then reacts with the FeS and yields calcium sulphide, which passes into the slag.

This explanation must be considered as rather artificial since with the greatest care exercised during the deoxidation process while the slag is entirely white, it contains nevertheless a small amount of metallic oxides. In this case the petroleum coke will deoxide without doubt iron and manganese oxides, but not lime.

It is far more probable that the removal of the sulphur gas is accomplished by the well-known reactions which occur in a blast furnace as well as in an open-hearth furnace, as follows:

$$2\text{CaO} + 2(\text{Fe or Mn}) \text{S} + \text{C}_2 = 2\text{CaS} + 2(\text{Fe or Mn}) + 2\text{CO}$$

$$2\text{CaO} + 2(\text{Fe or Mn}) \text{S} + \text{Si} = \text{A CaS} + 2(\text{Fe or Mn}) + \text{SiO}_2$$

In a Heroult furnace the removal of the sulphur takes place far more rapidly than in an induction furnace. This is easily explained without any artificial theory concerning the formation of calcium carbide in the furnace.

In the Heroult furnace the arc cuts through the layer of slag and the current passes through the metal along contact surface with the slag. An enormous amount of heat is produced in the slag and this causes an energetic reaction with the formation of calcium sulphide. This does not take place in the induction furnace because the slag is heated to a much smaller degree.

As to the Natusius furnace and the removal of sulphur in the same, it must be said that the high temperature produced by the three arcs in the slag causes as complete a desulphurization as in the Heroult furnace, even without the addition of petroleum coke and without any supposition of the formation of calcium carbide.

I am of the opinion that in all furnaces in which an arc enters the slag and in which the slag is heated by the passage of the current along the surface of contact between metal and slag, all reactions must be much more successful and much more energetic than in furnaces in which the bath is heated in a different way. In this respect the Natusius furnace is quite perfect. Besides the arcs which pass through the slag and the further passage of the current along the contact surface of metal and slag (as in the Heroult furnace) there is also a passage of current from the upper electrodes through slag and metal and heating them (as in the Girod furnace). Moreover, part of the energy is supplied from the bottom electrodes through the upper layer of the bottom into the bath, heating

the lower layer of the bath and especially the bottom hearth. Therefore, the whole metallic bath, the slag layer, and the bottom material are all heated at the same time.

This is a very important advantage of the Natusius furnace over other systems.

Further, with this system it is possible to connect the furnace directly to the central station without having to fear any sudden fluctuations of load which would damage the generators, since the slag layer, the steel bath, and the bottom material act like an electric buffer and counteract the load fluctuation due to short circuits of the arcs. For this reason the furnace does not need any regulation at normal running.

Finally, an additional transformer suitably chosen enables one to vary at will the energy supply to the bottom electrodes and to heat the metallic bath and the bottom to a higher temperature when required.

#### Reliability of the Furnace.

At first sight the bottom of the hearth, especially the mass rammed on the upper surfaces of the bottom electrode, seems to represent a weak point of the Natusius furnace. It might be feared that cracks might form and the bottom would be damaged more rapidly than the other parts of the lining. However, practice shows that these fears are unfounded.

The bottom requires not more attention and care than that of an ordinary basic open-hearth furnace.

While I was present at Friedenshütte it became necessary to stop operation in order to change the roof, which was burned through at one place; the furnace was not used from Friday to Monday. The roof was taken off and, after the furnace had cooled down, the hearth bottom was cleaned from slag. The bottom lining was in very satisfactory condition, although it had stood 445 furnace charges without any great repairs. This figure is taken from the works records. The condition of the hearth bottom would have permitted continuation of operation for quite a time and it was not yet possible to say when repairs would have become necessary.

The arched roof had burned through at one place while I was at the works and was repaired by putting on a brick plaster. But as there was no rush of work and as considerable heat was lost through this defective part of the roof, the operation of the furnace was stopped to change the roof. The roof which was then removed had stood eighty-six or eighty-seven charges. In general, the life of a roof is ninety to ninety-five charges, according to the works records.

The weakest point of electric steel furnaces is to be found in the carbon electrodes. The wear and tear of the carbon electrodes is quite considerable with the Natusius furnace in Friedenshütte as well as with the Heroult furnace in Remscheid and at the Obuchow works. For the Natusius furnace in Friedenshütte, however, the electrode consumption is somewhat reduced by the use of three-phase currents and the smaller cross-sections employed on that account. In the manufacture of carbon electrodes no brilliant success has yet been obtained.

#### First Cost.

The following figures on *first cost* relate to conditions in Germany.

A furnace of 2 or 3 tons capacity costs \$5,250.

A furnace of 5 or 6 tons capacity costs \$6,750.

A furnace of 8 tons capacity costs \$8,500.

A furnace of 12 tons capacity costs \$10,000.

These figures do not include the cost of the generator, transformer, cables, etc., nor the license fee.

Now it must be admitted that a refining cost of \$4.20 per ton is much too high. The reason for this unsatisfactory result is to be found in the unsuitable arrangement of the plant.

The electric furnace is at a considerable distance from the converters and is separated from them by a long open-hearth furnace plant with five furnaces. The ladle filled with molten converter metal must be carried by means of the crane through the whole open-hearth furnace plant where there is always

some delay so that much time is lost. Moreover, this ladle cannot be brought directly to the electric furnace but its contents must first be poured into another ladle in which it is brought to the electric furnace.

The cost of operation at the steel works in Friedenshütte is given in Table IV.

TABLE IV. COST OF REFINING PER TON OF STEEL.

	Kilograms of material used.	Price per ton of material in dollars.	Price per ton of finished electric steel.
Molten converter steel.....	1000	.....	.....
Ore (Krivoi-Rog) .....	25	7.29	0.18
Lime .....	30	3.	0.09
Sand .....	3	0.50	0.005
Fluorspar .....	4	6.55	0.0275
Petroleum coke .....	3	9.50	0.03
Ferromanganese (60%) .....	6	39.50	0.2375
Ferrosilicon (75%) .....	1	77.50	0.0775
Aluminim .....	0.5	335.	0.1675
Roof (\$55 life only 50 chgs.) .....	.....	.....	0.22
Magnesite powder .....	4	12.50	0.05
Fine clay .....	4	3.75	0.015
Dolomite mass .....	15	8.50	0.1275
Chamotte .....	4	4.375	0.015
Carbon electrodes .....	5.70	72.50	0.4125
Nipple .....	.....	0.515	0.03
Wages (six men, namely two melters, one ladle man, two slagmen and one man at the furnace doors) .....	.....	.....	0.45
Electric energy, (270 kw-hours at 0.625 cent).....	.....	.....	1.6875
Amortization (10%) and interest (5%) of \$25,000.....	.....	.....	0.375

Total cost of refining per ton of finished electric steel, \$4.20

For these reasons there are long intermissions between the different charges. These intermissions are generally one hour. In this time the furnace cools off and the next charge is delayed. The converter metal is charged rather cold into the furnace.

Further, completely finished converter steel is poured into the electric furnace after manganese alloys have been added to it for deoxidization; as already mentioned, the metal contains 0.4 to 0.5 per cent manganese which must be removed in the electric furnace with a great waste of time and energy. It would be much better to charge the electric furnace with converter metal before manganese alloys are added.

On account of these disadvantages only five charges and surely not more than six charges can be finished during a day, while with a well regulated operation and without any unduly cooling of the furnace and without delay in the transport of the metal, seven or even eight runs could be easily made per day. Under such conditions the cost of refining would be reduced to \$3.75 per ton approximately.

With such a low refining cost (which is chiefly a result of the low rate charged for electrical energy in Germany) it is possible to produce very pure steel economically. But it is necessary that the electric furnace be placed right near the converter plant and that it be designed for the same capacity as the converter, that is, if the converter has a capacity of 12 tons the furnace must have the same capacity. The design of a 12-ton Natusius furnace is given in Fig. 4.

Of all proposed electric steel furnaces the Natusius furnace must be recognized to be most suitable for simultaneously heating the slag, the bath, and the bottom, and for this reason as well as on account of the possibility of regulating the effect of heating from the bottom electrodes, the Natusius furnace is worth the careful attention of metallurgists.

St. Petersburg, Russia.

### The Use of Anti-Piping Thermit in Ingots.\*

BY DR. C. CANARIS.

Anti-piping thermit is used to improve the quality of steel ingots by reducing the following defects:

1. Shrinking holes and piping.
2. Injurious segregation in the top of the ingot.
3. Large blow holes.

Furthermore the number of the small blow holes is reduced and the resulting material made much denser.

The anti-piping thermit process is based on several patents belonging to the firm Th. Goldschmidt A. G. in Essen, of which the German patent No. 159,757 is the most important. The thermit used as anti-piping thermit in steel ingots consists principally of a mixture of metallic aluminum and iron oxide in equivalent parts. If this mixture is brought to a temperature of over 1200 deg. C. (2200 deg. F.) the iron oxide is reduced by the aluminium according to the well-known equation:



During this reaction a large amount of heat is set free, which brings the products of the reaction to about 3000 deg. C. (5400 deg. F.). The anti-piping thermit is applied in sheet iron cans as shown in Fig. 1. By means of a thin iron rod, inserted in the hole in the can (Fig. 2)—in most cases a scrap rod of about 10 mm ( $\frac{1}{2}$  in.) diameter is sufficient—the anti-piping thermit can is introduced into the liquid metal at the point where the reaction is to take place.

Some years ago many experiments with the anti-piping

thermit have been made at different plants, but to the best of my knowledge these were only tried on silicon-treated material, especially forging steel, and never to material not treated with silicon. The purpose of these experiments was to decrease the piping by introducing the thermit cans into the

heads of the ingots directly after pouring, allowing the reaction to take place there. It was thought that through the absorption of the heat produced during this reaction the heads of these ingots would be kept hot and the piping thus reduced. A. Obholzer has published a detailed report regarding these experiments. (*Stahl und Eisen*, 1907, July 31, page 117; Aug. 7, page 1155.) From this report it can be seen that the improvement caused by the application of the anti-piping thermit for heating the heads of the ingots is very small and not in proportion to the expenses involved. The reason is that the heat produced by the thermit reaction was altogether too small to increase the temperature of the ingot heads effectively and that extraordinarily large quantities of thermit would be necessary to produce sufficient heat.

The methods were, therefore, changed and the thermit can was introduced at a fixed time after the ingot had been filled. The can was pushed down as far as possible in the mould, preferably clear to the bottom, so that the reaction might take place at the lowest possible point. In this way, as explained hereafter, a strong mechanical effect is exercised on the liquid metal in the interior of the ingot. Special additions to the anti-

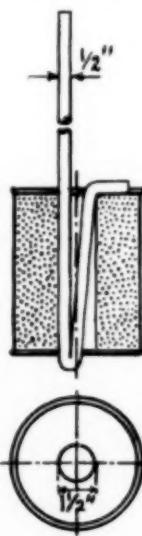


FIG 1  
APPLICATION OF ANTI-PIPING THERMIT.  
FIG 2

\*Translated from *Stahl und Eisen*, February 22.

piping thermit are made with the idea of decreasing the melting point of the slag, and thus entirely preventing particles of slag from remaining in the body of the ingots. This improved process was first applied only to silicon-treated material and good results were achieved in several steel works. Encouraged by these results I decided to use the anti-piping thermit also for connection with soft steel not treated with silicon.

As the results obtained were favorable from the first a systematic development of the process was carried out. On the basis of a great number of experiments made with ingots, weighing from 1 ton up, the quantity of thermit sufficient for each weight of ingot was accurately determined. It was then determined experimentally at what time, after pouring the ingot, the thermit should be introduced, and when more metal should be added in order to obtain the most favorable result. From these experiments, the following directions for this process may be laid down.

The pouring of the ingots is accomplished in the usual way, either by direct pouring from the top or by the more complicated bottom-pouring method. The filled moulds are allowed to rest until the solidification has so far advanced, that on the walls of the mould a crust of considerable thickness is formed. The required thickness of the crust depends, of course, principally on the size of the ingot. Then the anti-piping thermit can is pushed to the very bottom of the mould and is held there until the thermit reaction has taken place. The reaction results in a sudden and strong seething motion in the iron. After the reaction ceases, the surface of the still liquid material sinks, according to its condition, from 70 to 150 mm (3 to 6 in.), while the solidified walls remain intact. Then enough hot metal is quickly added, by direct pouring from the ladle, to bring the surface of the still liquid interior of the ingot to the same level as the solidified walls. The moulds are then immediately covered and the ingots left to cool.

The effect of the anti-piping thermit applied in this manner is as follows:

The products of the thermit reaction which are formed in the lower part of the ingot, namely thermit iron and slag, have a much higher temperature and are considerably lighter than the surrounding steel. Therefore, they are both driven with great force toward the top. This accounts for the boiling and seething of the steel in the mould previously mentioned. It is evident that this strong boiling has a favorable effect. It serves to mix thoroughly the entire fluid material and the iron at the top which has been cooled considerably by radiation is replaced by hot metal from the interior of the ingot. This tends to prevent to a great extent the formation of piping and is made still more effective by the large amount of new metal added. By this operation the reduction of volume caused through shrinkage and the removal of gases is compensated to such a degree, that very often the excess material is pressed toward the top with sufficient force to lift the cover. The formation of a pipe is, therefore, rendered entirely impossible. Furthermore, a large amount of gas is liberated as it is not necessary to cover the ingots prematurely, a practice generally followed in ordinary practice. A further removal of the gases is accomplished by the strong boiling of the material caused by the thermit reaction.

Where anti-piping thermit is not used, there is a chance that large and small gas bubbles will be caught in the ingot as the metal becomes pasty, thus causing blow holes and porous metal. With the anti-piping thermit reaction, however, all gases are thrown to the top. Furthermore any segregated and deleterious elements in the steel are likewise thrown to the top where they can do no harm.

These actions are favored by the fact that the escape of gas is not prematurely interrupted by the covering of the mould, and in consequence of the addition of a quantity of fresh metal, the head of the ingot stays hot for a considerable length of time. Therefore, one is able to avoid with certainty the piping and occurrence of large blow holes. A material

extremely dense and free from gases is obtained. For these reasons the thermit ingots yield a very high output in the rolling mill. For example, plates rolled from thermit ingots can be used practically in their entirety; only small parts of the top and bottom of the sheets have to be scrapped. As the effect of the anti-piping thermit, if applied properly, is absolutely certain, one can figure when using thermit ingots in plate mills on an essentially higher output than when using ordinary ingots.

Scrap plates, which under normal conditions constitute a pretty large percentage, are almost eliminated. The total output is essentially increased; the expenses, however, which are incurred through the use of anti-piping thermit, are relatively small.

In Table I the various sizes of the cans, their price and the weights of the ingots are given:

TABLE I. SIZES OF THE THERMIT CANS FOR VARIOUS INGOT WEIGHTS		
Size of Can	Contents	Ingot weight metric tons
00	0.65 kg. 1.43 lbs.	1 to 1.7
0	1.3 kg. 2.86 lbs.	1.7 to 3.5
1	2.5 kg. 5.50 lbs.	3.5 to 8.0

Up to the present time I have had no opportunity to treat larger ingots with thermit. However, the firm of Th. Goldschmidt A. G. has provided larger cans for such ingots.

The cost of the scrap rods, used for the introduction of the cans, is almost equalized through the iron resulting from the thermit reaction and taken up by the ingot.

Below are given a few results of the many experiments which were undertaken to establish the effect of the anti-piping thermit. In the first experiments the anti-piping thermit was applied to improve the quality of metal not treated with silicon. From heat 1143 of the July 12, 1911, two ingots of about 5800 kg (12,800 lb.) each were poured. One of the two ingots was treated with thermit in the above manner, while thermit was not used in the second ingot.

The treated ingot was rolled to a plate of 2927 mm (9 ft. 7 in.) width, 19 mm (3/4 in.) thick and 9850 mm (32 ft. 3 in.) long. The output amounted to 75 per cent. A test piece taken from the bottom of the plate gave 35.7 kg per square millimeter (50,800 lb. per square inch) and 31 per cent elongation; one from the top, 37.6 kg per square millimeter (53,500 lb. per square inch) and 29.5 per cent elongation; the lap at the top of the plate was 250 mm (10 in.).

The ingot not treated with thermit was rolled to a plate 3070 mm (10 ft. 1 in.) wide, 23 mm (7/8 in.) thick and 6340 mm (20 ft. 8 in.) long. The output was 68 per cent. The tensile test piece from the bottom showed 35.3 kg per square millimeter (50,200 lb. per square inch) and 33 per cent elongation; while the tests from the top of the plate showed 38.6 kg per square millimeter (54,900 lb. per square inch) and 25 per cent elongation. The lap was 600 mm (24 in.) long.

Further results are given in Table 2 (p. 234); the ingots mentioned were poured from the bottom in one lot. The analysis shows the great uniformity of the thermit-treated material. Table 3 shows a few analyses of the results with the corresponding tensile tests.

TABLE III. ANALYSES AND TENSILE TESTS OF THERMIT PLATES

Plate	Analysis				Corresponding Test Pieces			
	C%	Mn%	P%	S%	C%	Mn%	P%	S%
1	0.082	0.40	0.022	0.040	0.078	0.40	0.019	0.038
2	0.072	0.49	0.048	0.046	0.070	0.48	0.033	0.040
3	0.076	0.45	0.028	0.020	0.070	0.43	0.026	0.018
4	0.071	0.47	0.040	0.030	0.069	0.45	0.032	0.026
5	0.082	0.45	0.028	0.020	0.074	0.43	0.026	0.018

Plate	Tensile Strength in lb. per sq. in.	Elongation %	Tensile Strength in lb. per sq. in.	Elongation %
1	53,893	28.0	51,476	30.5
2	53,609	29.5	51,476	32.0
3	54,320	27.5	51,192	31.5
4	52,329	28.5	49,912	32.0
5	54,889	26.5	51,334	31.0

A great many experiments of this kind were made and always with the same results. In no case was any extensive

piping found and the segregation was comparatively small, although the plates were used with the least possible amount of cropping. These favorable results led to the treatment of a great number of ingots of various weights with the anti-piping thermit in order to get a good average result.

During the time from July 12 to October 31, 1911, 4409 ingots, weighing from 1 to 8 long tons, were treated with anti-piping thermit. The cost of this treatment amounted to 35 cents per ton of ingots. All ingots treated with anti-piping

Furthermore, the moulds used in this process have to be more conical in shape than usual, and in consequence of this the Harmet ingots yield a very high percentage of scrap, from the edges of the plates. This practically eliminates all the advantages gained by this process. Regarding the effect of the pressure, I do not think that any transference of the segregation in the ingot to the top takes place.

The anti-piping thermit acts in a manner all its own and is much superior to the Harmet process in every respect. For

Remarks With Thermit	Plate 1	Ingot weight lbs. 2827	TABLE II. RESULTS OBTAINED WITH ANTI-PIPING THERMIT					Place where test piece was taken
			Width inches 94.20	of plate inches 28/64th	Length of lap in inches 6.29	Tensile strength lbs. per sq. in. 52,400 55,100	Elongation per cent. 32.0 28.5	
" "	2	2849	84.00	28/64th	4.72	52,100 53,800	30.0 28.0	Base Head
" "	3	2849	94.20	28/64th	4.72	51,400 54,300	32.0 29.5	Base Head
" "	4	2838	94.20	28/64th	4.72	51,800 54,200	31.0 29.5	Base Head
Without Thermit	5	2739	84.60	25/64th	30.31	52,300 59,600	28.0 20.0	Base Head

thermit were rolled. These ingots were in all cases 3 to 5 per cent lighter than those ordinarily used, for the same sized plates. Of the 4409 ingots rolled to plates only sixteen showed any defects, which could be traced to a faulty quality of the material. The total weight of the ingots from which these sixteen plates were made, amounted to 22½ long tons; the total weight of the 4409 ingots was 7628 long tons. The loss was thus less than 0.3 per cent, which is extremely low. The total increase of output was more than 5 per cent of the quantity charged.

Table IV gives the weights of the treated ingots and the cost of the applied thermit.

TABLE IV.

The total weight of 4409 ingots from 391 heats was 7628 long tons they originated  
Cost of treatment

No. of ingots	Weight ingots on long tons	Mks.
2134	1.0 to 1.5	2869 anti-piping cans with No. 00 = 5451.10
1200	1.5 to 2.0	
532	2.0 to 2.5	
206	2.5 to 3.0	1302 anti-piping cans with No. 0 = 4296.60
157	3.0 to 3.5	
72	3.5 to 4.0	
70	4.0 to 4.5	238 anti-piping cans with No. 1 = 1511.30
23	4.5 to 5.0	
6	5.5 to 6.0	
9	7.5 to 8.0	
4409		Total mks. 11259.00 = \$2,815

If we figure the difference between the price of the plates and the price for the scrap at \$15 per ton, the use of thermit ingots for the plate mill will permit a saving of  $76.28 \times 5 \times 15 = \$5,720$ , while the steel works only expended \$2,815 for thermit. The net gain amounts to \$2,905 or about 38 cents per long ton.

Added to the above saving are other great advantages which cannot be put down in figures, namely the increase of the production without enlarging the works and the decrease of the resulting scrap. It ought to be further considered that on account of the almost entire absence of scrap plates the work in the rolling mill is much more uniform, as there is no delay in the work and in the filling of orders due to cutting and cleaning of scrap plates. Finally the undoubted improvement in the quality of the steel should not be lost sight of.

The anti-piping thermit process has shown itself as an extraordinarily simple and effective help in improving the quality of soft steel ingots untreated with silicon. Whether it can also be used for silicon-treated metal and just how it is to be applied in such material, experiments which are now being conducted, will show.

Up to the present time only the Harmet pressing process has been used on a large scale for material untreated with silicon. For the application of this process a very large investment is required; and it is so complicated that only a small part of the total production of a mill can be treated.

its introduction no investment is necessary. Its application is very simple and its effect certain. The cost of the anti-piping thermit is low and the benefit of its use can easily be figured in dollars and cents.

Duisburg-Wanheim, Germany.

#### Liquefied Natural Gas.

The Bureau of Mines has just issued Technical Paper No. 10, entitled "Liquefied Products from Natural Gas; Their Properties and Uses," by Irving C. Allen, and George A. Burrell. The authors endeavor to show how natural gas, which is being allowed to escape almost without restraint in almost all of the petroleum fields of the country, may be conserved.

The authors in outlining their investigations, say: "By fractionating natural gas, either during or after liquefaction, four products can be commercially obtained. Roughly, these four products may be described as follows: (1) The gaseous product, the common natural gas of commerce; (2) the semi-liquid product, known as the new 'wild' product, which should be used only as a liquefied gas and should be held in high-pressure steel containers only; (3) the light liquid product, or light gasoline used for blending with heavy naphthas; and (4) the heavy liquid product, or ordinary high-grade gasoline.

"The possibility of handling the second product in the way that Pintsch and Blau gases are handled, enabling small towns, hotels and country estates to have the advantage of gas illumination, manifestly opens a new field of comparatively great importance in the natural gas industry and should add materially to the investments made in the so-called 'natural gasoline' industry.

"The liquefaction of gases by pressure is not a new industry, but only recently has its application to natural gas been recognized as practicable.

"Up to the last two years the general practice in the manufacture of liquid natural gas was to make the product by compression of the gas in single-stage compressors operated at a pressure of 150 to 300 lb. per square inch. The one product obtained, so-called 'natural gasoline'; was run into a tank and 'weathered.' The weathering consisted in allowing the lighter portions to volatilize spontaneously and escape into the open air until such time as the boiling away of the liquid had practically ceased. Thus the process involved a loss of 25 to 50 per cent, or even more. This loss was an absolute waste, not only of power and of cost of operating the engines and compressors but of the product itself.

"The next step in the industry was to pass the waste gases (of which only the small quantity used for power had been utilized) from the single-stage compressor through a higher-stage compressor, thereby getting a second and more volatile

product—a 'wilder' liquid—which was run back into the first and mixed with the first or heavier condensate. This mixture was then again weathered to a safe degree, whereby it lost the greater part of the more volatile product that had been condensed in the second stage.

"Recently the process had been improved another step, in that the first stage compressor product is run into one tank and handled as ordinary gasoline; the second compressor product is run into a second tank and handled as a lighter gasoline, with which the heavy refinery naphthas can be enriched or enlivened.

"The last-mentioned method of using the second stage compressor product should receive wide recognition, and a market for the product should develop that would be no mean factor in the industry. Blending in the proportions of, say 1 part of the product to 4 or 5 parts of the refinery naphthas makes these heavy naphthas more volatile and of greater value as fuel for automobiles; it also greatly increases their general usefulness. The proportions to be used in blending, however, must be determined more definitely by test.

"The natural gas of this country frequently contains light products that do not condense in the second-stage compressor, and for which it is practicable and necessary to install three, four, and even higher stage compressors. These light products—true gases at ordinary temperatures and pressures—can be compressed and liquefied, but the liquid gases so obtained must be handled as gases and not as oils.

"The mistake heretofore made in the 'natural gas gasoline' industry, as some have recognized, has been the attempt to handle the light gaseous products as oils and not as gases. Until the manufacturers of this lightest third or fourth stage compressor product recognize its gaseous nature, the absolute necessity for insuring the safety of the public involves certain restrictions in its transportation, and not until the realization that this extremely volatile liquid should be handled only in strong steel containers capable of withstanding high pressures will it be transported with safety."

Copies of this paper may be obtained by writing to the Director, Bureau of Mines, Washington, D. C.

### Concentration of Ore Slime by Centrifugal Separation.

It has long been recognized that the great deficiency in ore dressing methods lies in slime concentration. The invention of the Wilfley table and the subsequent development of many similar types of reciprocating concentrators, may be said to have given us high efficiency in the separation of pulps ranging in size from 12 to 60 or 80 mesh, but equivalent progress in slime concentration has not been evident.

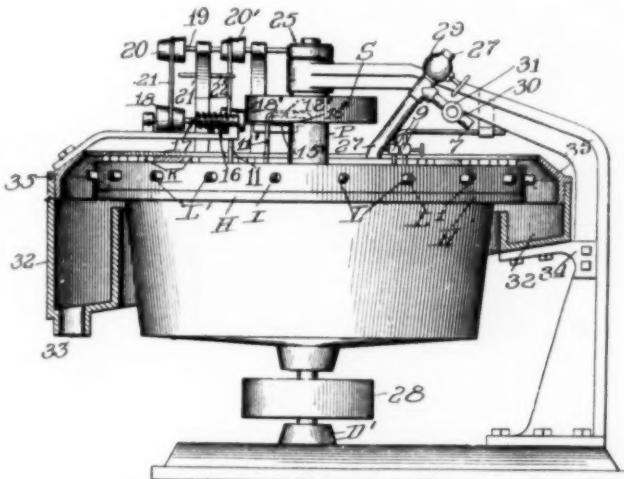
Inventors of successful concentrators of coarse pulps have turned their attention to the problem of slime concentration, but they have usually worked along the lines which have proved successful in coarse concentration, and the machines thus evolved can hardly be said to have filled entirely the requirements of treating pulps as fine as 150 and 200 mesh.

The belt vanner and the stationary canvas table also are in use as slime concentrators, but their capacity is low, and thousands of square feet of concentrating surface is required for even moderate efficiency. In fine, the great demand for the successful slime concentrator has not yet been met, and consequently the ore dressing industry continues to sustain a loss which, in the aggregate, is enormous.

In our issue of January, 1912, page 3, editorial mention was made of "another and more logical way" in which the slime problem was being attacked by the Anaconda Copper Mining Company, at Anaconda, Mont., and the Cananea Consolidated Copper Company, at Cananea, Mexico. The statement had reference to the experimental tests of the Peck centrifugal ore separator, patented by Philip F. Peck, of Chicago, Ill.

Although no data have been made public regarding these

tests, it is understood that the centrifugal machine has proved more successful in handling slime at these places than has any other form of mechanical device. The difficulty of the problem will be understood from the fact that the pulp is finer than 200 mesh. Although it contains as much as 2.75% copper, it has been almost worthless, and has been impounded anticipating the day when some successful means of treatment would be found. It is reported that the Peck centrifugal con-



into this space, where the heavy mineral becomes bedded on the inner surface of the outer cone, which is moving more rapidly than the inner cone. The lighter, or gangue slime rises to the top of the space, and is discharged through small openings in a number of agate cylinders into an annular launder. When the bedding operation is complete, and the gangue slime or tailing has been removed from the concentrating space, a slight water wash is automatically admitted for the purpose of removing a middling product from the surface of the concentrate; and finally, when the middling has passed out into its launder, a flood of water is admitted to scour out the concentrate which is similarly discharged into a separate laund-

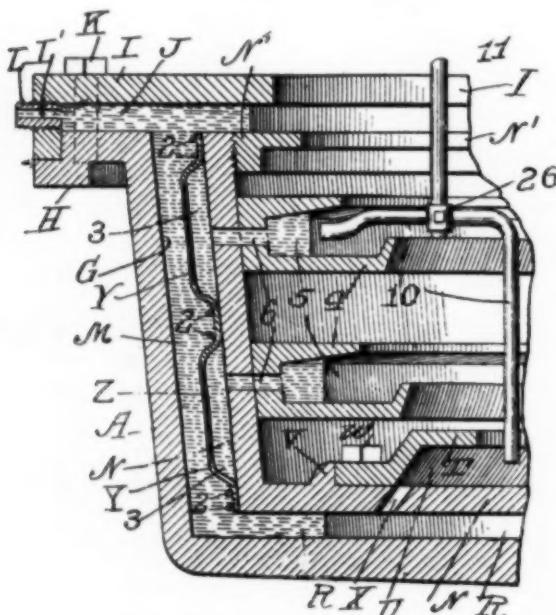


FIG. 3.—DETAIL OF SEPARATING PASSAGE SHOWN IN FIG. 2.

der. During the period of discharging the concentrate the speed of the outer cone is reduced to that of the inner.

Thus the operation is cyclic; and all the phases, such as feeding the pulp, discharging tailing, middling and concentrate, etc., are automatically controlled by a set of cams on a mandrel driven by an independent motor. The annular launder also is automatically raised and lowered to correspond with each phase of the cycle, so that the different products are discharged into separate compartments thereof.

The space between the two cones is kept constant during the successive phases of a cycle by means of an elastic medium encircling the inner cone. This elastic medium is adapted to be distended by water while the bed of concentrate is forming, and can be contracted as the concentrating space fills. The admission and discharge of water for this purpose is automatically controlled to correspond with the cycle through which the operation extends.

Apparently the phases of a complete cycle will vary according to the material treated, depending, say, on the ratio of concentration; but once the duration of a cycle is properly determined for any pulp, the various phases may be automatically controlled.

With this general idea of the process in mind, attention may be directed to more specific description of the machine as given in the patent papers. Reference will be made to different figures in order to describe different parts which may be shown plainly in those figures; but it will be understood that a completed machine is supposed to embody the parts described, even though those parts do not appear in each figure.

Figs. 1 and 2 represent respectively a general view and central vertical section of one type of centrifugal separator. Referring to Fig. 2, the separating surface is shown at G, being the inner surface of a vessel A which is closed at the bottom

and substantially open at the top. It is rigidly attached to a central shaft C. A ring I (better shown in Fig. 3) is attached to the vessel A in such a manner as to form an annular space J. The depending part of the ring I is provided with a row of screw-threaded holes into which are screwed agate plugs L having openings L', for the discharge of the various products of concentration. A launder 32 surrounds the machine in a position adapted to receive the discharged products. In this figure the launder is shown without separate compartments which appear in other figures.

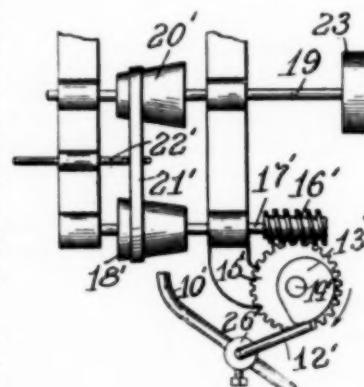
Within the vessel A is located a second vessel N, some details of which are more clearly shown in Fig. 3. The outer wall of the vessel N is concentric with the inner wall of the vessel A; and between these two surfaces is the concentrating passage M. The inner vessel is attached by hub O to a sleeve P mounted rotatably on the central shaft C. Thus the inner and outer vessels may be rotated independently and differentially as desired, the shaft pulley for the outer vessel being shown at 28, and the sleeve pulley for the inner at S.

The inner vessel embodies, in addition to the rigid part N, a flexible member Y (Fig. 3) made of rubber or similar fabric, securely and appropriately fastened to the inner vessel. The object of this flexible covering is to provide means of controlling the width of the separating passage between the two vessels, in order that the surfaces may be maintained in comparatively close proximity. For example, when the feed is first admitted into the machine, it is desired to have the separating passage narrow to facilitate the bedding of the concentrate; but it is also desirable to widen the separating passage as the bed of concentrate thickens on the surface G. Therefore means are provided for periodically expanding the flexible member by liquid pressure, and alternately contracting it by removing the liquid.

In Figs. 2 and 3 are shown the details whereby this contraction and expansion is obtained. The inside of the inner vessel is provided with annular troughs 4 adapted to receive liquid into their channels 5, through pipes 8. Around the troughs are bored holes 6 connecting the channels with the expansion chambers 3, so that when the machine is in operation water will be driven by centrifugal force into the expansion chambers 3, and distend the flexible member toward the concentrating surface G.

As stated before, it is advantageous that the expanding pressure on the flexible member Y be diminished gradually as concentrate accumulates in the separating passage. Wherefore, scoop conduits 10 are supported on rods 11 as shown, and adapted to be moved by cams 13 into or out of contact with the surface of the water in the channels 5. These scoop conduits may thus be used to remove water from the expansion chamber, allowing the latter to contract at the proper time. The water thus removed may flow downward into the feed compartment U and join the feed, or may be forced upward through a pipe and out of the machine as shown in another design. A top plan view of the cam arrangement is shown in Fig. 4. The cams are rotated by worm gears controlled by cone pulleys driven from the central shaft C; although it is understood that in the machines now in operation, these cams and all other means of controlling the pulp, water, launders, etc., are operated independently of the means used to revolve the two cone-shaped vessels.

With this description of the parts in mind, the operation of



the machine during one cycle will be understood, as follows: The outer vessel A is revolved at a desired rate of speed, which is greater than that at which the inner vessel N revolves. The flexible covering Y of the inner vessel is expanded by water so that it approaches within a short distance of the surface G, in order to cause a liquid scouring or washing of the separating surface, or of the concentrate which later accumulates there.

Slime pulp is fed into the machine through the pipe 27, the

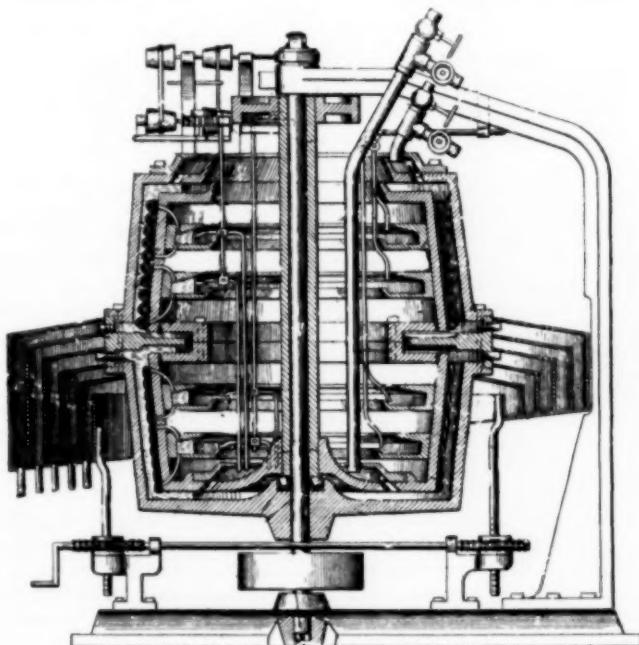


FIG. 5.—MODIFIED FORM OF CENTRIFUGAL CONCENTRATOR.

flow being controlled at the valve 29. It passes into the annular feed chamber U, thence through passages X, and outward and upward into the passage M. As the operation proceeds and concentrate begins to bed on the separating surface, the scoop conduits 10 are moved by their cams so that they gradually remove water from the channels 5, and allow the flexible member Y to contract. This continues until a bed has formed thick enough practically to fill the separating passage. During the bedding process, the tailing has been rising through the separating passage, flowing out of the machine through the openings in the plugs L into the annular launder.

The flow of feed is then cut off at the valve 29, the speed of the outer vessel is reduced somewhat, while that of the inner is maintained, and clear water is introduced through a branch pipe 30, controlled by valve 31. By this time the cams have moved the scoop conduits so as to allow a further accumulation of water in the channels 5, which distends the flexible covering Y. Thus the clean water scours the bed of concentrate, removing a middling product which passes out through the same openings L<sup>1</sup>. Finally, the scouring and washing action is increased, and the bed of concentrate is washed out of the machine and collected in the launder. This completes a cycle of operation. As stated before, the means for governing the various phases of a cycle are automatic in their action, and independent of the means used to rotate the separating vessels.

Fig. 5 illustrates another patented type of separator, which is simply a combination of upper and lower separators having the same cycle and discharging through a common series of openings. This figure shows the launder compartments for receiving the different products of the machine, and means (not automatic) for raising and lowering the launder to bring the proper compartment into juxtaposition with the openings.

Fig. 6 illustrates a combination which includes automatic means for controlling the phases of a cycle, such as introducing

pulp and wash water; admitting and removing compression water from the flexible chamber; and raising and lowering the launders to permit the different products to be collected separately. The arrangement appears rather complicated, but it must be remembered that there are several parts which must work in phase in order that the cyclic operation of the machine can continue uninterruptedly. Although this figure shows the governing means to be actuated by the same power that drives the separating vessels, it is understood that in the working machines the different phases of a cycle are controlled from an independent source of power.

In Fig. 6 the launders are moved into the desired positions by means of the wedges 114, which are operated by the hydraulic piston 119, which, in turn, receives water from the 4-way

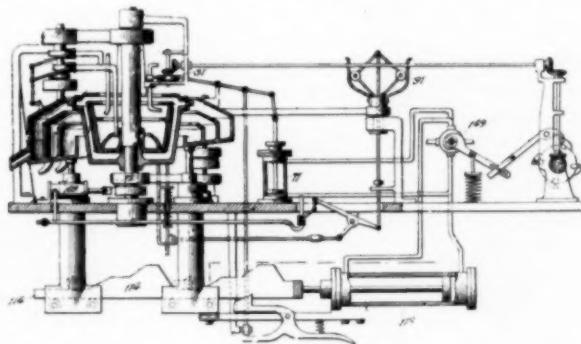


FIG. 6.—DETAIL OF MECHANISM GOVERNING AUTOMATIC REGULATION OF CYCLES OF SEPARATION.

valve 149. The feed of slime pulp and wash water is controlled by the lever 146 attached to the launder. The cams governing the scoop pipes which remove compression water from the channels leading to the expansion chamber are shown at 31, and are actuated by the hydraulic piston 71, which also receives water from the 4-way valve 149. The action of the hydraulic piston 109, which is controlled by levers attached to the governor 91, moves a brake 75 which, at the proper time, reduces the speed of the outer vessel.

The foregoing description gives, in brief, the principles on which the Peck centrifugal slime concentrator operates, although the actual mechanical means of accomplishing the operation may be different from those given. The fact that the machine has been tested with apparent success leads to the hope and belief that it may prove to be the slime concentrator for which there is such a demand; and the publication of authentic details of actual practice will be awaited with interest.

**Vanadium and Dynamic Steel Tests.**—*American Vanadium Facts*, Vol. II, 1, just issued by the American Vanadium Company of Pittsburgh, Pa., contains a quite interesting article on mechanical tests of steel. In these days of alloy steels a determination of the maximum breaking strength and percentage of stretch at rupture no longer suffice, but "dynamic" tests become important. Woehler, in the middle of the last century, devised a method which showed that metals will fail from fatigue or gradual fracture when subjected to repeated stresses well within their breaking strength. Prof. Arnold in Sheffield devised a modification of Woehler's test. It was intended as a shortened Woehler test to determine the same property of steel as is determined by the latter. But it was found that Arnold's test predicted liability to fracture from causes altogether apart from Woehler's fatigue phenomenon, and registered potential brittleness, which the Woehler test not only failed to detect, but emphatically pronounced absent. The whole matter, while generally of great importance for testing and designing engineers, is especially interesting in connection with vanadium steel, as vanadium is now looked upon as the element par excellence which greatly increases the elastic limit and dynamic strength of steel without impairing its ductility.

## Radiation Pyrometry.

By G. A. SHOOK.

### I. Radiation Laws.

There are today four precise laboratory methods for measuring high temperatures.

*Electric-resistance Pyrometer.*—In this pyrometer use is made of the variation of the electric resistance of metals with change of temperature. Since resistance can be measured with extreme precision, the method permits of very precise measurements of temperature up to 1000° C.

*Thermoelectric Pyrometer.*—It utilizes the variation of the e.m.f. with temperature developed at the junction of two dissimilar metals. This pyrometer may be used for temperature up to 1600° C., when the thermo-couple consists of wires of platinum and platinum-rhodium or iridium.

*Total Radiation Pyrometer.*—In this type of pyrometer the total radiation from hot bodies is taken as a measure of their temperature. This instrument requires a device for determining very small changes in temperature, and does not admit of very great accuracy, but is very convenient for very high temperatures.

*Optical Pyrometer.*—In the case of pyrometers of this class, temperature estimation is made by means of a photometric comparison for a particular wave length, between the radiation from one standard lamp and the radiation emitted from the body under observation. This is a very precise method and is available for the highest known temperature.

### TEMPERATURE SCALES.

Temperature is generally measured by the magnitude of some physical change produced in a body by heat, such as the expansion of mercury in the ordinary mercury-in-glass thermometer, change of electrical resistance, in the resistance thermometer, or the production of an e.m.f. in the thermo couple.

In order that the different thermometers may be comparable, they are generally standardized by means of invariable and easily reproduced fixed points, and the two universally employed are the temperature of melting ice and the temperature of boiling water. The interval between the fixed points, may be subdivided into any convenient number of divisions, each of which will represent one degree of temperature.

In the centigrade scale, the zero is arbitrarily taken as the melting point of ice, and the interval divided into 100 parts.

Clearly the magnitude of a division will depend upon the substance employed and the temperature change from 1 to 2 may not be the same as the temperature change from 50 to 51. The zero point on the centigrade scale, does not, of course, mean that there is no heat present when the mercury stands at zero.

Since the expansion of air is about twenty times as great as that of mercury, some form of an air thermometer is generally used for accurate scientific purposes. From the law of Charles, the volume of a given mass of gas, under constant pressure, increases by a constant fraction of its volume at zero for each rise of temperature of 1° C. This law leads to an absolute scale of temperature. According to Boyle's law, the product of the pressure and volume of a gas at constant temperature, is a constant. A combination of these two laws gives the "gas law"  $PV=RT$ , where T=absolute temperature (°C+273°). The absolute zero is 273° below the arbitrary 0°C.

Until recently, all "temperatures" in precise work where referred to the international Normal Scale of the constant-volume hydrogen thermometer.

It must be remembered, however, that these gas laws are not rigorously true for real gases, such as CO<sub>2</sub>, H, and N, so that different gas scales will differ slightly among themselves, and furthermore, no one gas is satisfactory throughout the entire range of temperatures which are used in gas thermometry.

Hydrogen can be used for very low temperatures, but above 300° C it is unreliable. Nitrogen, on the other hand, cannot be used for low temperatures, but is suitable for high temperatures. In the absence of a perfect gas, we have practical

standard gas thermometers, such as hydrogen and nitrogen, for which thermodynamic corrections have been determined. In practice, however, the gas thermometer is never employed, by reason of the difficulties inherent in its use, and furthermore, because there are numerous other thermometers more convenient which can be compared with the gas thermometer.

A theoretical thermometer scale, independent of any substance used, has been worked out and it is known as the "thermodynamic scale." It can be easily shown that this scale agrees with the perfect gas scale. In exact work, it is necessary, therefore, to define temperature in terms of the thermodynamic scale rather than the "normal" or "gas scale." Especially is this true in the case of radiation pyrometry where the laws and formulas developed, have their foundation on thermodynamic laws.

### RADIANT ENERGY.

There are three distinct modes of transference of heat, namely conduction, convection, and radiation.

In conduction, heat is transferred from particle to particle of a body as in the case of solids. The conduction of heat along an iron bar is a familiar example.

In convection, heat is taken up by matter and carried with it in its motion. It can be shown experimentally that in the case of liquids and gases, which are poor conductors, heat is transferred by the motion of sensible masses of matter.

Heat may also be distributed as radiant energy, just as light is propagated. While it was formerly supposed that luminous bodies give rise to three distinct kinds of radiation; luminous, heat, and actinic, it is now universally known that the three are identical. There is no simple theory of radiation which will explain all the phenomena satisfactorily. The electron hypothesis will, perhaps, give us the best physical conception.

In order to account for the more complex phenomena that are constantly appearing in physics and chemistry, we must imagine that the chemical molecule or atom consists of a nucleus of positive electricity in equilibrium with particles of negative electricity or "electrons." The mass of an electron has been found to be about the 1/1700th part of the mass of the hydrogen atom. As long as an atom remains intact, it is electrically neutral, but when an electron escapes from it, it becomes positively charged.

Now we imagine that any charged body has "electrical lines of force," emanating from it, and terminating on some other body of opposite charge, and, further, that a moving charge, such as an electric current in a metallic conductor, maintains a constant magnetic field about it. This magnetic field changes in direction and magnitude as the velocity of the moving charge changes. Now, while the theory is very complex, it is easy to see that when an isolated moving charge, with its lines of force emanating in all directions, is suddenly brought to rest, some sort of a disturbance must be propagated along the lines of force. This disturbance is most effective in a direction perpendicular to the motion of the moving charge.

Let  $e$ , Fig. 1, represent an electron with its electric lines of force, at rest, and  $e'$  the same electron moving to and fro in the direction BB'. A wave disturbance will be sent out along the lines of force in the second case. It will be a maximum along AA' and zero along BB'. It is this electromagnetic pulse which produces the sensation of heat or light.

If this is true we should expect a magnetic field to produce some effect upon spectroscopic lines, that is, we might expect the "D" line of sodium to broaden, or to shift its position slightly. This magnetic effect has been found by experiment, and this experimental fact is one of the foundation stones of the electron theory.

The manner in which this pulse is produced, affects the quality of the radiation markedly. In the X-ray tube, the free negative cathode particles on being suddenly stopped by a metallic plate give rise to the well known Roentgen Rays. In this case the pulse propagated along the lines of force is very abrupt, because the velocity of the moving charge is abruptly

brought to zero. The moving electron within an atom or from one atom to another does not suffer such a sudden change of velocity and therefore the resulting pulse is much smoother and of longer duration.

Now according to the electron theory, the optical properties of substances are closely related to their electrical properties. A substance which conducts electricity is called a conductor, and one which does not, an insulator, but of course, there is no sharp dividing line. All insulators, in fact, will conduct to some extent at very high temperatures so that in this sense the term "metal" is often applied to any substance which is in any degree a conductor. In the case of metals, electrons are constantly escaping from the atoms, leaving the latter charged. The electron will not leave the metal, however, except at high temperatures. These escaped electrons are in turn being attracted toward positive atoms. It is this change in the electron's

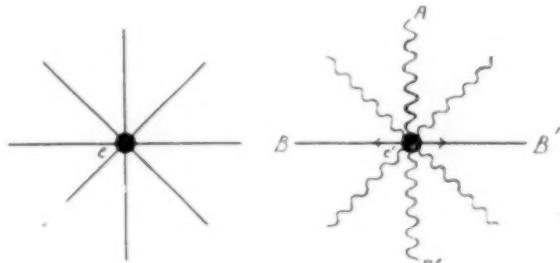


FIG. 1.—ELECTRON AT REST AND IN OSCILLATION.

velocity, brought about by its impact with atoms, which gives rise to the radiation that is propagated throughout space.

According to the dynamical theory of heat, any hot body is composed of minute particles, identical with the chemical atom, which are in rapid vibratory motion. When heat is given to a body, the velocity of these particles is increased and its temperature is raised. However, it is the motion of the electrons into or out of the atom, that produces radiation and not the motion of the atom itself. We know of no other radiation except electro-magnetic. Furthermore, if all atoms could emit radiation then any material substance would soon disintegrate, due to the motion of its atoms. In a solid body, the electron may penetrate for a small distance into the atom, and then move away in a different direction, so that the pulse is not so abrupt as in the case of Roentgen rays.

The disturbance sent out from incandescent solids at high temperatures, produces what is known as "white light," which may be resolved into a continuous spectrum (i. e., a continuous band of colors from violet to red) by means of a prism. The energy thus radiated in the form of transverse etherical waves is transformed into heat, whenever the wave disturbance falls upon some material body.

The impinging of these waves upon the retina of the eye, produces the sensation of red light, when the radiating body is about  $600^{\circ}$  C. Violet light corresponds to vibrations of small periods or short wave lengths, while red light corresponds to long wave lengths.

The visible spectrum occupies a very small part of the entire known spectrum, which extends from wave-lengths of about  $0.0001$  cm. to  $0.03$  cm. A unit commonly used to express wave lengths is  $\mu = 0.0001$  cm. The visible spectrum extends from about  $0.7\mu$  to  $0.4\mu$  (Fig. 2).

The radiation sent out by cooler bodies is quite similar to that emitted by incandescent bodies, but the maximum part of the energy is in the region of the long wave lengths, which are not visible to the eye. Nevertheless, it must be remembered, that even at room temperature, radiation corresponding to all parts of the spectrum is being emitted from bodies, only to a much smaller degree.

There are several optical methods for determining "wave-lengths" in the visible part of the spectrum, and the wave-lengths of the most useful emission lines of such elements as

mercury, hydrogen, etc., have all been carefully determined. These may be used for calibrating an ordinary prism spectrometer. The wave length of an unknown line may then easily be determined. By means of such a calibrated spectrometer, the mean wave length transmitted by a piece of colored glass may be determined.

When light or heat falls upon any body, it may be reflected, transmitted, or absorbed, or all three phenomena may occur simultaneously. A body which absorbs all the energy, incident upon it, such as lampblack, neither transmitting nor reflecting any, is said to be perfectly black. The emissive power of a body is the ratio of the energy per square centimeter that it emits at the particular temperature to the energy per square centimeter that a black body would emit at the same temperature. Kirchhoff formulated the law that the emissive power,  $e$ , of a body equals the absorptive power,  $a$ . For a black body, then,  $a = e = 1$ .

The curves in Fig. 2 represent the distribution of energy of a black body for various temperatures. It is seen, from these

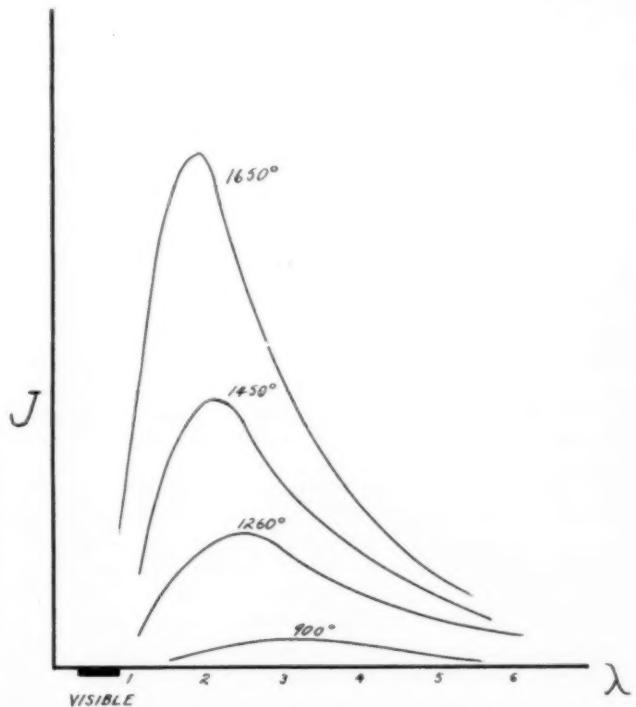


FIG. 2.—DISTRIBUTION OF ENERGY RADIATION FROM A BLACK BODY.

experimentally determined curves, that for low temperatures the energy maximum, i.e., the highest point of the curve, is far out in the invisible red. Furthermore, the maximum shifts toward the shorter wave-lengths as the temperature is increased. Now as a black body radiates all the energy incident upon it, or all that it absorbs, since  $e = 1 = a$ , no other body at the same temperature can radiate more energy than a black body. This is true for any wave-length, as well as the whole spectrum.

A black body, however, would not make as efficient a luminous radiator as some body with an abnormal emission in the visible spectrum and a corresponding suppression of radiation in the infra red. In the case of the Auer (Welsbach) mantle there is practically no emission between  $4\mu$  and  $7\mu$ .

Again it is seen from the above curves that at high temperatures the relative amount of radiation in the visible part of the spectrum is much greater than at lower temperatures, i.e., the energy increases with temperature much faster in the short wave-lengths than in the long ones. It is for these two reasons that metallic-filament lamps, such as tungsten and tantalum, which can be worked at high temperatures, are more efficient than the carbon lamp. Carbon is approximately a

black body, but a carbon lamp cannot be worked at high temperatures. Platinum, on the other hand, departs considerably from a black body, but it would make a more efficient lamp than any of the metallic-filament lamps if it could be worked at high temperatures.

There is no known substance which is perfectly black, but it can easily be shown that the interior of a uniformly heated enclosure radiates like a black body.

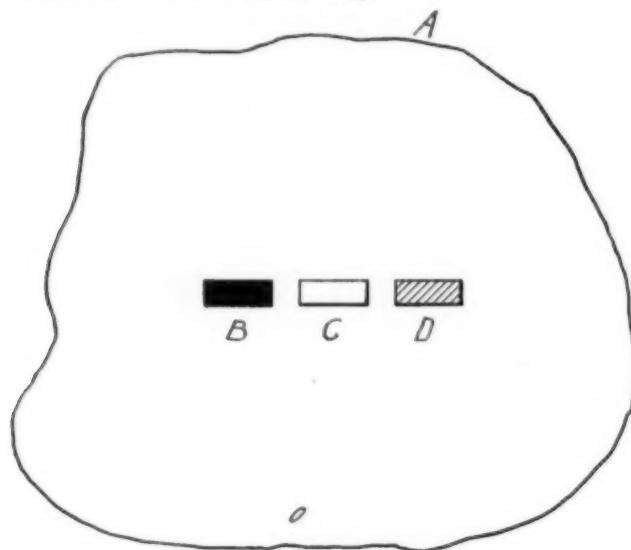


FIG. 3.—BLACK BODY.

Let *B*, *C* and *D* (Fig. 3) represent three bodies of different emissive powers, which are completely surrounded by an envelope *A* maintained at a constant, uniform temperature. We will now suppose that the entire system has come to temperature equilibrium, so that every point within *A* is at the same temperature. Hence *B*, *C* and *D* have the same temperature.

To make the case perfectly general we will make some further assumptions, namely, that *B* is a black body, such as a piece of carbon, therefore its absorptive power is unity; *C* is perfectly transparent, so that its absorptive power is zero, and *D* absorbs one-third the radiation incident upon it, reflects one-third and transmits one-third, i.e., its absorptive power is one-third.

Now, any body placed within *A* will radiate a certain amount of energy; part of it may be transmitted through it, part may be reflected from its surface, and the rest the amount it emits that is proper to itself. For example, if its absorptive power is one-fourth, that means that of all the energy falling upon it, only one-quarter is absorbed, and this one-quarter alone produces heat within the body. The other three-quarters is reflected from the surface, or transmitted through it, so that all the energy it receives it again radiates. We are here considering only the radiation that is purely thermal; that is, chemical effects, such as luminescence, must be excluded.

Let us now consider the relative radiation of the three bodies; that is, let us see how they would appear to an observer at *O*. Since *B* is in temperature equilibrium with the walls of *A*, it must radiate to them as much energy as it receives from them. If it radiated more than it received it would cool down, and if it radiated less it would become hotter than *A*, which is contrary to our assumption that *B* and *A* are at the same temperature. *B* will therefore radiate to the wall behind it as much as it receives from that wall and consequently to an observer at *O*, *B* would appear just as bright as the surrounding wall. Hence the inside wall of *A* will radiate as a black body. Since *C* transmits perfectly, the observer would, of course, see the wall beyond and therefore *C* will radiate as a black body. Now *D* only radiates one-third of the energy it receives, but it allows one-third of the energy falling upon

it, from the rear wall, to pass through, and it also reflects from the front wall one-third the incident energy, so that it will radiate as much energy as *A*, *B* or *C*, and will therefore be as bright as *A*. Hence *D* radiates like a black body.

Now, all of the energy radiated by *B* is proper to itself because it absorbs everything and emits just as much as it absorbs. All of the energy radiated by *C* comes from the wall of *A*, and none is proper to itself, since it absorbs nothing. Since all three bodies and the interior wall of *A* all radiate like a black body, it follows that any body with any emissive power whatsoever, within a uniformly heated enclosure, made of any material whatsoever, would radiate as much energy as an ideal black body at the given temperature. To an observer at *O* the interior of *A* would appear uniformly bright and no contour of objects would be perceptible. If all the bodies could be quickly removed from *A* and examined in a darkened room they would present quite different appearances. *B* would appear as bright as it did when within *A*, but *D* would only appear one-third as bright and *C* would radiate nothing whatever and therefore appear black. This is a matter of experience and can be easily verified experimentally. This conception of a black body is very important in the application of pyrometry.

#### LAWS OF BLACK BODY RADIATION.

Stefan deduced from experiment, and Boltzmann deduced from thermodynamic considerations, the law that the total radiant energy emitted from a black body is proportional to the fourth power of the absolute temperature, or

$$J = KT^4$$

where *K* is a constant.

Another important law is Wien's "distribution law," which gives the relation between energy and absolute temperature for a particular wave-length, i.e., a particular color of the spectrum. This law does not hold, however, except for high temperatures.

It is as follows:

$$J = C_1 \lambda^{-5} e^{-\frac{C_2}{\lambda T}} \quad \dots \dots \dots \quad (1)$$

*J* = energy corresponding to wave-length  $\lambda$ .

$\lambda$  = wave-length measured in  $\mu$ .

*T* = absolute temperature =  $273 + t$ .

*C*<sub>1</sub> and *C*<sub>2</sub> = constants.

*C*<sub>1</sub> = 14500 when  $\lambda$  is measured in  $\mu$ .

*e* = base of the natural system of logarithms.

Formula (1) may be written in the form

$$\log_{10} J = K_1 - K_2 \frac{1}{T}$$

where

$$K_2 = C_2 \frac{\log e}{\lambda}$$

hence

$$t = \frac{K_2}{K_1 - \log_{10} J} - 273 \quad \dots \dots \dots \quad (2)$$

This equation, which has been considerably simplified, gives the relation between the temperature *t* in degrees centigrade and the logarithm of the energy and may be applied to any pyrometer using monochromatic light, in which the luminous intensity can be varied in a continuous and determinate manner, as in the Wanner and Le Chatelier.

*K*<sub>2</sub> can be calculated when  $\lambda$  is known and *K*<sub>1</sub> is an arbitrary constant depending only upon the units of *J*. When *J* is known for some particular value of *t*, *K*<sub>1</sub> may be calculated from equation (2).

The methods of applying these two equations to radiation pyrometers will be taken up in a subsequent paper.

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## Notes on Chemistry and Metallurgy in Great Britain.

(By our Special Correspondent.)

### The Growth of Cast Iron.

In a lecture to the Sheffield branch of the British Foundrymen's Association on the 20th February, Professor Carpenter gave the results of his further researches on the growth of cast iron after repeated heatings which had been undertaken with the view of providing a remedy. He found that the substance mainly instrumental in producing the growth was silicon, but the presence of carbon in the form of graphite was necessary before growth could occur. Graphite gave rise to channels through which the gases penetrated and oxidized the silicon; and both with superheated steam and with furnace gases at high temperatures the process of growth was identical. He suggested manganese as a preventive, and, after an extensive series of experiments, was in favor of the use of an alloy containing 1.5 per cent of manganese, 2.5 per cent of carbon, and 0.5 per cent of silicon, the melting point of which was 1345 deg. C.—practically the same as that of a No. 1 hematite containing rather over 4 per cent of total carbon. The sulphur and phosphorus in his test bars were only 0.01 per cent, but the raising of their figure to 0.06 per cent would not make any material difference, would put the metal in the usual grade of English irons, and reduce the melting point to but little over 1300 deg. C.

### Electric Smelting.

The progress of the electric furnace in Britain is slow yet steady, and the cost of power supply is the chief obstacle to be overcome before a more rapid advance can be looked for; but that the advance will ultimately take place appears to be beyond question.

At Sheffield Messrs. Vickers, Ltd., are building an 8-ton Héroult furnace of the three-electrode type, which will be the first of its kind to employ three-phase currents in England. The three-phase furnace permitting the use of static transformers with an efficiency of fully 97 per cent should effect a considerable reduction of power cost per ton of steel as compared with a single-phase furnace with a motor generator set with intrinsically much lower efficiency.

The Electric Furnaces and Smelters, Ltd., who have three large furnaces in operation at Luton producing ferro-alloys, chiefly ferro-tungsten, propose to erect two more furnaces each of which will have nearly three times the capacity of the furnace they are now using. In the production of tungsten-steel the 50 per cent ferro-tungsten is preferable to one much richer in tungsten on account of the lower temperature necessary for its solution and the freedom of the steel from hard spots, flaws, etc., resulting from imperfect solution of tungsten. The 50 per cent ferro-tungsten is a true solution of tungsten in iron, while the alloy with a higher tungsten content is a mere mixture of tungsten with such solution; and the undissolved excess of tungsten necessitates raising the steel bath to a considerably higher temperature, and even then is liable to cause defects.

### Smelting of Tin.

Very satisfactory results are reported from Cornwall where about a year ago a trial furnace, with a capacity of some 2 tons of ore per day, was erected for the smelting of tin. This work has been described in detail in the issue of September, 1911, of this journal (vol. IX, page 453).

### Sheffield Steel and Smoke.

The anti-smoke nuisance agitators are moving strongly just now at Sheffield and are urging the adoption of gas furnaces for reheating; but a considerable number of manufacturers say that smoke is necessary in certain processes.

Professor Arnold energetically supports the manufacturers and asserts that the skilful use of smoke by experienced workmen is largely instrumental in maintaining the famous quality of Sheffield steel. The carbon quality of the steel, he says, de-

pends on having a smoky flame in the reheating furnaces; and it is a matter of the highest importance, because otherwise the surface of the steel would be deprived of the carbon which confers specific properties and would be comparatively useless. The oxidation which would remove surface carbon would, moreover, probably penetrate and cause rotteness in the ingot under the hammer and in the rolls. Steel must not be reheated in an oxidizing or "dry" flame, but with a smoky or "green" flame; and the professor is strongly opposed to the use of gas furnaces because the completeness of the combustion of the gas results in a "dry" flame which does not yield any deposit of carbon. Steels, and particularly hard steels, must be heated in a smoky flame giving a black carbon deposit, and the emission of a dense black smoke is a necessary accompaniment of the early stage of reheating; while as the operation proceeds the regulation of the furnace to avoid excess of oxygen in the flame also renders further evolutions of smoke unavoidable; and the higher the carbon content of the steel is, the greater will be the quantity of smoke required.

M. W. F. Beardshaw, at the annual meeting of the Sheffield Chamber of Commerce which took place early in February, said Professor Arnold deserved the thanks of the manufacturers for the able explanation of the subject he had given to the public. They did not want to make an unnecessary quantity of smoke, but the trade must be protected against interference. Good Sheffield steel stood first in the markets of Europe, and any action which might be prejudicial to the quality of that steel would be disastrous to the industry.

### Trade Notes.

The activity in the steel trade in Scotland is indicated by the announcement that the Clydebridge steel works are to be reopened. About four years ago, when the demand for steel was not equal to the supply, these works were closed by an agreement with the Scottish Steelmakers' Association which guaranteed the shareholders 5 per cent on the capital on condition that no steel should be produced for market purposes.

New steel works to have an output of 5000 tons per week are to be put down at once at the Burrows, Swansea, at a cost of between £100,000 and £150,000, by Messrs. Baldwin & Company, who now have extensive tinplate works there. The subsequent erection of blast furnaces is contemplated.

The Knowles Oxygen Company, Ltd., is erecting new works at Bromborough Port, on the Mersey, to be in operation by next October, for an output of 200,000,000 cu. ft. per year of oxygen and hydrogen. The plant consists of two gas engines of 1750 b.h.p. each coupled directly to D.C. generators, and the requisite Mond gas and recovery plant. It is expected that oxygen will be sold direct from the holders at prices ranging down to 5/- per 1000 cu. ft., and compressed in cylinders at £1 per thousand; while the purity guarantee will be raised to 99½ per cent.

### Engineering Imports and Exports.

The returns issued by the Board of Trade for January, 1912, show a satisfactory all round improvement on the figures for the same month last year. Imports of iron and steel, including manufactures, reached £1,069,593, an increase of £244,288; and exports were £4,354,238, an increase of £523,551. Imports of other metals, including manufactures, totalled to £2,754,249, an increase of £506,748; while exports were valued at £956,866, an increase of £91,878. Imports of electrical goods amounted to £139,996, an increase of £14,191; and exports are put down at £275,091, an increase of £72,095. Machinery was imported to the amount of £514,947, and exported to the amount of £2,666,642; showing increases of £81,228 and £180,559 respectively. Imports of new ships were only £57—a decrease of £9734; and exports, value £183,222, decreased £46,986.

### The Coal Miners' Strike.

The threatened national strike of the coal miners is an accomplished fact, and about 1,065,000 men have ceased work.

The English owners, although unable to concede the unqualified minimum wage for all workers were prepared to accept the principle wherever the conditions of work reduced a man's output. The Welsh miners have unjustifiably broken their unexpired contract; and the failure of any approach to an arrangement in their case is not to be wondered at. There are rumors that the Premier has threatened to introduce a Minimum Wage Bill in the House of Commons next Monday unless the owners give way; but any such measure would not be welcomed by the law-abiding section of the general public, amongst whom the feeling is growing that the refusal of the English miners to accept the fair offers of the owners deprives them of any claim to sympathy. The duration of the strike cannot be very prolonged, for the total of the men's union funds is little if anything over £2,000,000; but even if it last only a fortnight or less its effect on the trade of the country will be disastrous. Iron and steel works, tinplate works, potteries, and factories generally, have already begun to close, and soon millions of workers will be without employment. All trades must suffer severely; and engineering and chemical industries will have to bear a heavy share of the evils resulting from the dissemination of socialistic ideas among the classes whose scanty educational opportunities make them the easy dupes of the professional agitator.

#### Market Prices.

FEBRUARY, 1912.

*Copper* has shown an upward tendency steadily throughout the month. Starting at just under £62 it had dropped nearly a pound by the 6th, but afterwards rose steadily, and finished at £64.7.6.

*Tin* has been confined to fairly narrow limits for this market. Starting near £195 it has kept within £3 either way, and closes at £199.

*Lead*. Inclined to rise. Opened £15.18, closes £16.3.9.

*Haematite* rose fairly well after the 9th. Opened 62/2; closed 66/-.

*Scotch Pig* has kept fairly steady, on or about 55/-, closing at 55.6.

*Cleveland* has been rather stronger at 49/- to 49/6, closing 49/6½.

*India-rubber* has been fairly quiet. Opening at 4/8 it closes at 4/7½ per lb.

£ s. d.

Alum, lump, loose, per ton.....	5.15.0
Antimony, black sulphide powder, per ton.....	20. 0.0
Borax, British Refined Crystal, per ton.....	16.10.0
Copper Ore, 10 to 25% per unit.....	10/10½ to 11.4½
Copper Sulphate, per ton.....	22.17.6
Carbolic Acid liquid, 97/99%, per gal.....	1.8
Creosote, ordinary good liquid, per gal.....	2
Camphor, 1 oz. tablets.....	1.9½
Caustic Soda Ash, 48% ordinary, per ton.....	5.10.0
Hydrochloric Acid, per cwt.....	5.0
Mica, small slab, original cases, per lb.....	6d. to 2.0
Naphtha, solvent 90%, 160 deg. C. per gal.....	8½
Petroleum, Russian Spot .....	8
Quicksilver, per bottle.....	8. 7.6
Sal Ammoniac, lump, first delivered U.K. per ton....	42. 0.0
Sulphate of Ammonia, f. o. b. Liverpool, per ton... .	14.18.9
Sulphur, recovered, per ton.....	5. 0.0
Shellac, standard, T.N. Orange spot, per cwt.....	4. 4.0
Tin Ore, 70% per ton.....	£124 to 126. 0.0
Zinc, Vieille, Montagne.....	30. 0.0
Platinum .....	9. 5.0

The differences in price are therefore as follows:

#### Higher.

£ s. d.

Copper, ton .....	2. 7.6
Tin .....	4. 0.0
Lead .....	5.9

Haematite .....	3.10
Scotch Pig .....	6
Cleveland .....	6½
Quicksilver, bottle .....	2.6
Ammonia Sulphate .....	5.0
Tin Ore .....	4. 0.0

#### Lower.

£ s. d.

Copper Sulphate, ton.....	17.6
Carbolic Acid, gal.....	1

The impending coal strike will naturally influence the price of metals.

#### Synopsis of Current Literature.

##### Gold and Silver.

**Cyaniding at the Talisman Mine, New Zealand.**—Some interesting details of tube milling, agitation and filtration are given in an extensive paper on the Talisman mine by Arthur Jarman, in the *Proceedings of the Australasian Institute of Mining Engineers*, Sept. 30, 1911.

The author is not clear as to the grade of ore treated, but from some data given in connection with the mining operations we gather that the ore is comparatively high grade, with a high, but varying, ratio of silver to gold. Part of the ore is oxidized, but occurrences of sulphides of copper, lead and iron are common.

The general plan of mill treatment is as follows: Crushing in Blake crushers and stamps; classification of the stamp product, the coarse sand being re-ground in tube mills and combined with classified overflow; classification of the tube mill discharge, with return of coarse portion to the tubes; amalgamation of fine pulp from tubes; concentration on vanners; leaching of sand tailing and agitation of slime; vacuum filtration; precipitation by zinc shavings; production of bullion.

The tube mills were not part of the original plant, but were installed later. The following grading analyses show the effect of the tubes in grinding.

Mesh.	By Stamps.	By Stamps and Tubes.
+30	0.08	0.00
+40	3.24	0.00
+50	12.37	0.03
+60	6.63	0.30
+80	14.76	10.45
+90	9.68	....
+100	0.67	6.32
-100	52.56	....
+150	....	12.46
+200	....	13.19
-200	....	57.24

Two of the tube mills are lined with H. P. Barry's patent sectional honeycomb liner, and one with F. C. Brown's iron plate and rib liner. An opportunity was thus afforded to compare the two under identical conditions. From the point of view of the operator, the Brown liner is better, for should a rib or plate break or loosen, a small leak promptly starts at the bolt holes, thus giving timely warning. With the Barry liner, however, the contents of a quarter sector may be dislodged without the defect being discovered until the mill is opened. The time required for renewing liners is also in favor of the Brown. The following grading tests on the tube mills shows the work of each kind. In studying the table it must be remembered that the tube mill "feed" is composed of the "battery pulp" plus the returned sand from the classifiers following the tubes. The "plate heads" are the overflow from these classifiers.

Mesh.	Battery pulp.	Plate heads.	—Feed—		—Discharge—	
			Barry.	Brown.	Barry.	Brown.
20	7.6	0.02	2.0	2.6	0.36	0.42
40	17.8	0.18	12.3	13.4	7.9	9.0
60	11.4	2.2	24.6	25.7	25.3	25.9

80	8.8	9.6	26.6	28.6	31.0	32.7
100	5.4	11.8	9.6	9.1	9.9	9.1
120	3.0	5.8	3.6	3.1	4.0	3.9
150	3.0	5.6	2.8	2.1	3.2	2.5
Thru 150	42.7	64.3	18.2	14.7	17.5	16.2

The classifiers used are of the diaphragm type, having a  $\frac{1}{2}$ -inch space around the diaphragm to insure an even flow of pulp. The feed to the tubes averaged practically 1:1 of moisture and dry pulp.

Amalgamation is done on ten plates each 4 ft. 9 in. wide by 12 ft. long. About 28 lb. of mercury is used every morning in dressing the plates, and the loss sustained is 0.25 oz. per ton of ore crushed. The percentage of recovery by amalgamation has been as follows: 1906, 37.3; 1907, 39.1; 1908, 40.1; 1909, 40.7; 1910, 40.3 for 6 months prior to installing tube mills; and 31.0 for 3 months following installation of tubes. The reduction is ascribed to a reduced grade of ore.

The concentrate recovered on vanners averages 1.25% by weight and 15 to 18% of the gross value of the ore treated. The ratio of gold to silver in the bullion varies from 1:3 to 1:30.

The tailings from the vanners are sent to leaching tanks. Butters and Meins distributors are used, and the tanks are full of water before pulp is turned into them. The overflowing slime is treated by agitation. As the ore is variable in gold and silver content, the treatment of any tank is not decided upon until it has been sampled and assayed. The average treatment for a charge of 1.40 tons of ordinary grade ore is as follows: Allow to drain completely, and then give an alkaline wash of 4 tons medium solution with 200 lb. soda ash, followed by 4 tons more of medium solution and 4 tons strong solution. The displacement is then continued by adding 8 tons strong solution with 270 lb. NaCN, and repeated lots of 10 tons of strong solution until the effluent shows 1.1% total cyanides (0.7% free KCN). The percolator cock is then closed and the solution circulated by a small air injector, thus keeping the solution in continuous circulation during contact. After 4 or 5 days contact the pulp is sampled and is usually shown to be ready for the next step. The solution is then drained off, displaced by 5 tons medium solution, followed by washes of from 80 to 120 tons of weak solution. The residues are then of sufficiently low grade to be sliced to the river.

Owing to the variable silver content the strength of solution during contact has to be varied. The following examples give a good idea of the practice on rich, medium and low grade ore:

#### GOLD.

Tons Charge.	Head.	Tail.	Ext. %
140	\$21.21	\$1.21	94.5
138	11.42	0.79	93.1
145	4.04	0.42	89.7

#### SILVER.

Head.	Tail.	Ext. %.	Lb. NaCN	= KCN per Ton.
10.61 oz.	1.69 oz.	84.1	320	2.65
5.50	0.61	89.1	270	2.27
2.41	0.34	85.8	130	1.04

The slime is agitated by air at 40 lb. pressure in tanks 12 ft. diameter and 30 ft. high. The solution has a strength of 0.19 to 0.35% KCN, and the time is 24 hours, after which the pulp is filtered by vacuum filters of the fixed frame type. The filter tanks are filled with pulp which is kept agitated by means of air. The solution is drawn by two wet, double-acting vacuum pumps, maintaining a vacuum of 23 to 25 inches of mercury. The cake is allowed to build up to a thickness of  $\frac{3}{4}$  to  $1\frac{1}{4}$  inches in an hour. A weak solution wash is then applied for an hour and a half to two hours, after which the cake is discharged with 18 to 25% solution as moisture. Separate zinc boxes are provided for the precipitation of strong, medium and weak solutions.

**The Specific Gravity Flask Used for the Measurement of Pulp.**—The specific gravity flask should prove a most

useful and indispensable instrument for efficient and reliable control of the conditions of mill pulp. At least this is the opinion of H. STADLER as expressed in the *Journal of the Chem. Met. & Min. Soc. of S. Africa*, November, 1911. He gives five different applications of the instrument, together with formulae.

I. Determination of Specific Gravity of Solids.—If the material is dry, a known weight in grams is placed in the flask, moistened and shaken with water to remove air bubbles, and water is added to fill the flask to the top or mark. This is weighed and the sp. gr. obtained by the use of formula 1. Wet material such as mill pulp is preferably dried and weighed after the determination of the sp. gr. of the pulp.

II. Determination of Specific Gravity of Pulp.—Almost all factors determining the nature of pulp from the millman's point of view are governed by the sp. gr. of the pulp. If the metric system is used the sp. gr. is at once obtained by dividing the net weight of the pulp by the volume occupied, or when a 1000 c.c. flask is used, by pointing off the last three figures of the net weight of the pulp in grams. The sp. gr. of slime pulp in which all solids are freely suspended may be measured by the hydrometer.

III. Determination of Percentage by Weight of Water and Dry Solid.—The usual method of determining these quantities is tedious and unsatisfactory, besides being unreliable and inaccurate. The author's suggestion is shown on formula 2.

IV. Tonnage Measurements of Total Pulp (Tp) and Dry Solids in Pulps (Ts).—The usual inconveniences of drying and weighing time samples of flowing pulps are avoided by reckoning the tonnage from the volume occupied by the total pulp (in cu. ft.) and its sp. gr., by formula 4 for the total pulp, and formula 5 for the dry solid portion only. Run the pulp into a box of known capacity, large enough to take at least one minute's run of the stream. Determine the sp. gr. from several representative samples taken simultaneously, and the calculated weight of pulp or dry slime passing during the time ( $t$  in seconds) is then used to give the hourly or daily quantities.

If, in all measurements, boxes of equal capacity are used, say 16 cu. ft. (the volume of half a ton of water) all the constant factors may be condensed into one. For a box of this capacity the formulae of Rand ore are:

$$\text{Tons of total pulp in 24 hours (Tp)} = \frac{43,200}{t} \cdot \frac{(p-1) 68609}{t}$$

$$\text{Tons of dry solid only (Ts)} = \frac{(p-1) 68609}{t}$$

Since the capacities of launders, tanks, pumps, etc., as well as the rates of overflow of classifiers are determined by the volume of the pulp, the fluid ton of 32 cu. ft. is frequently used as a unit. The tonnage of pulps of known sp. gr. are readily converted into fluid tons by the formulae 7 and 9, for total pulp and dry solids respectively.

V. Relations of Flowing Quantities in Classifiers.—The relation of flowing quantities in classifiers may be calculated from the percentage of any of their components measured at the inflow, overflow and underflow. No component can be measured so quickly as the percentage of dry solid taken with the sp. gr. flask. From the percentages of dry solid so obtained the tonnage of total pulp underflow (as a percentage of the total pulp inflow) is calculated from formula 10, and the tonnage of dry solid underflow (as a percentage of dry solid inflow) by formula 11. All the formulae are based on the sp. gr. of pure water, but they are applicable to ordinary cyanid solutions with enough accuracy for practical purposes, except in the case of thin pulps, such as turbid water, overflow of classifying tanks, etc. The author states that the sp. gr. of ordinary cyanid solutions assaying 0.126% KCN and 0.01% alkalinity has been determined to be 1.0021, and this small variation from pure water is almost negligible.

Formulae.—The values of the different terms in the formulae are as follows:

$v$  = content of flask in c.c.

$w$  = net weight of ( $v$ ) c.c. pulp in grams.

$p$  = sp. gr. of pulp (mixture of solid and water).

$d$  = sp. gr. of solid.

$T_p$  = tons of pulp (water and solid).

$T_s$  = tons of dry solid in the pulp.

$W$  = % by weight of moisture in pulp.

$S$  = % by weight of dry solid.

$V$  = volume of pulp in cu. ft.

$$k = \text{a constant for any particular value of } d = \frac{100d}{d-1}$$

$Fl$  = fluid tons ( $1 Fl = 32$  cu. ft. pulp).

#### I. Determination of density ( $d$ ) of material (a) in grams.

$$(1) d = \frac{a}{v + a - w}$$

#### II. Relation between sp. gr. ( $p$ ) and net weight of pulp ( $w$ ).

$$(2) p = \frac{w}{v}, w = v p.$$

#### III. Relation between % of dry solid ( $S$ ) and moisture ( $W$ ), and sp. gr. of pulp ( $p$ ). $W S 100\%$ .

$$(3) S = \frac{(p-1) k}{p}, p = \frac{k}{k-S}$$

#### IV. Tonnage measurements of pulps ( $T_p$ ) and dry solid in pulps ( $T_s$ ).

$$(4) T_p = .03125 Vp.$$

$$T_s = V \left( p-1 \frac{k}{3200} \right)$$

$$(5) T_s = V \left( p-1 \right) .04963.$$

Tonnages and fluid tons.

$$(6) T_p = Fl, p.$$

$$(7) Fl = \frac{T_p}{p}$$

$$T_s = Fl \left( p-1 \right) \frac{k}{100}$$

or when

$$\frac{k}{100} = 1.588 \text{ (Rand ore)}$$

then

$$(8) T_s = Fl \left( p-1 \right) 1.588$$

or

$$(9) Fl = \frac{T_s}{\left( p-1 \right) 1.588}$$

#### V. Relations of flowing quantities in classifiers.

##### Inflow (i)

$$\begin{aligned} Wi & \% \text{ water} \\ Si & \% \text{ dry solid} \end{aligned} \left. \begin{array}{l} \text{pi sp gr of inflow} \\ \text{po sp gr} \end{array} \right\}$$

$$Ip \% \text{ tons pulp inflow (100\%)}$$

$$Is \% \text{ tons dry solid inflow (100\%)}$$

##### Overflow (o)

$$\begin{aligned} Wo & \% \text{ water} \\ So & \% \text{ dry solid} \end{aligned} \left. \begin{array}{l} \text{po sp gr} \\ \text{Op tons pulp overflow} \end{array} \right\}$$

$$Os \% \text{ tons dry solid overflow (in \% of inflow)}$$

##### Underflow (u)

$$\begin{aligned} Wu & \% \text{ water} \\ Su & \% \text{ dry solid} \end{aligned} \left. \begin{array}{l} \text{pu sp gr} \\ \text{Up tons pulp underflow} \end{array} \right\}$$

$$\left. \begin{array}{l} Us \text{ tons dry solid underflow (in \% of inflow)} \\ \text{pulp (Tp)} \\ Quantities of underflow \frac{\text{dry solid (Ts)}}{100 (Si - So)} \end{array} \right\} \text{in \% of inflow}$$

$$(10) Up = \frac{Su - So}{100 Su (Si - So)}$$

$$(11) Us = \frac{Si (Su - So)}{pu \times 100 (pi - po)}$$

$$(12) Up = \frac{pi}{pu - 1} \times \frac{pu - po}{100 (pi - po)}$$

$$(13) Us = \frac{pi - 1}{pi} \times \frac{pu - po}{pu - po}$$

**Slow-Speed Chilian Mills.**—In a paper published by the Mexican Institute of Mining and Metallurgy Mr. J. B. EMPSON presents many arguments in favor of the slow-speed Chilian mill as a fine grinder. His work supplements very admirably that of Mr. D. C. Bayldon, who presented similar matter before the Institute of Mining and Metallurgy (London), and which was reviewed in our issue for February, 1911.

Mr. Empson is of the opinion that engineers generally do not appreciate the good qualities of this mill as a fine grinder, and that they follow too closely in the path of custom which dictates the use of other grinders. We quote from Mr. Empson:

"The types with which the writer is familiar are extremely crude both as regards the method by which the power is applied, as well as the impossibility of preventing the grit entering the bushing in which the trunnion shaft turns, hence causing an altogether needless expense in the upkeep. The mills in use at La Union in Pachuca, and at El Bote, Zacatecas, are of a different type to those above mentioned and are far superior in every respect. The main reason, I understand, why these latter types of mills have not come into more general favor is that of first cost, which, if my information is correct, would appear excessive; probably owing to their great weight, the question of freight and duties plays an important part."

"There are several claims to be made in favor of the Chilian mill in Mexico as compared with stamps, especially in small installations.

"People who argue in favor of stamps base their claims on the following or similar points: simplicity, ease with which repairs can be made, small space occupied, operation in units, availability of men who are familiar with their operation. Much can be written in favor of stamps, especially as applied to ores which do not call for reduction to slime, but on which leaching of sand is possible; under these conditions, where ore is to be crushed to  $\frac{1}{4}$  in. mesh in stamps and reduced to 30 or 40 mesh in tube mills, the slow speed Chilian does not compete. The slow speed Chilian is essentially a fine grinder and as such can show a mechanical efficiency that cannot be obtained by stamps and tube mills. Hence its application to Mexican conditions, where the latest metallurgical practice is to grind silver ore so that all will pass 100 mesh and in some mills so that not more than 20 per cent will remain on a 200 mesh screen. To do this work with stamps entails the installation of one tube mill to each 5 stamps when each stamp weighs 1250 pounds and putting these at their maximum capacity of 10 tons per stamp according to Mexican practice gives results as follows:

"10 stamps. 1250 lbs. each. 100 drops per min = 27 h.p. or 2.7 per stamp 100 tons deducting 10 tons for slime in batteries = 90 tons, divided between two tube mills gives a charge of 45 tons per mill. Then allowing two  $5 \times 16$  ft. mills to do this work the rated running power required for each mill would be 46 hp., totalling 92 hp.

"Therefore 27 hp. plus 92 = 119 hp. or 1.19 per ton.

"Tube mills require approximately 1 hp. for every ton ground to 200 mesh.

"To do the same work with slow speed Chilian mills would require an installation of:

"Four Chilian mills of similar design to those in use at S. Luis de la Paz requiring 15 hp. each and crushing approximately 100 tons or 25 tons per mill per 24 hours to an estimated degree of fineness as per figures published by Mr. Megraw on work done at the Sta. Elena mill at San Luis de la Paz.

On 60 mesh.....	1.15 per cent
Through 60 on 80 mesh.....	5.65 per cent
Through 80 on 100 mesh.....	0.15 per cent
Through 100 on 200 mesh.....	25.75 per cent
Through 200 mesh on.....	67.30 per cent

"Considering these figures it is seen that 34.70 per cent is the amount that will not pass a 200 mesh screen, and of this amount 25.75 per cent passes the 100 mesh screen and remains on 200 mesh, leaving the small percentage of 6.95 as staying on 100 mesh. It is to be regretted that no 150 mesh screen was used in this screen classification. This is stated to be an average result. Hence it is seen that in crushing 100 tons of ore by this method the amount to be reground is 34.75 tons, of which only 6.95 tons can be classified as being coarse. To do this work would take a 4 x 16 ft. tube mill using 27 hp. at rated running, making a total of 87 hp. for the whole installation, equivalent to 0.87 per ton of ore crushed to 200 mesh. Naturally the finer the product given to the tube mill to crush the greater will be the output of the mill. From experience gained from an installation of this kind it has been shown that the difficulty is to give the tube mill enough work to do in order to allow it to work efficiently. From the figures as given above it will be seen that with the Chilian mill installation there is a saving of 0.32 hp. per ton as compared with stamps and tube mills.

"Some extremely interesting comparisons have been made recently by writers in the technical mining journals as to the merits of stamps and tube mills as compared with slow speed Chilian mills.

"I would refer members interested to read with special attention the article by Mr. W. H. Urbiter published in the *Engineering and Mining Journal* of August 5, 1911.

"Mr. Urbiter in reviewing the situation gives figures as follows:

#### Sta. Elena S. Luis de la Paz.

#### 2 roller-type slow speed Chilian mills.

Tons per hp. Day.	Relative Mechanical Efficiency.
67.3 per cent through 200....	1.84      49.09 per cent
Hacienda La Union, Pachuca.	

Same as above.

80 per cent through 200 mesh.	1.70	46.13 per cent
South Africa. Heavy Stamps.		

#### Tube Mills.

90 per cent through 90 mesh..	1.53	40.48 per cent
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"Putting the South African practice on a par with Mexican practice, it would appear that if the South African mills were crushing to the same fineness as those in Mexico the relative mechanical efficiency would be much lower.

"American practice of 3 roller fast running type of Chilian mill:

39.4 per cent through 200 mesh.	2.35	55.73 per cent
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"Owing to the relatively small amount remaining on 200 mesh, it will be seen that this class of mill, when crushing to the same degree of fineness as the mills in Mexico, would lose to a large extent their relatively high mechanical efficiency.

"From the above figures it will be seen that when the fineness of the final pulp is taken into consideration, it is evident that the slow speed Chilian mill is a much more efficient machine to reduce ore to the required fineness as demanded by Mexican practice than any of the other combinations.

"Itemized costs per ton of ore ground, as published by Mr. Megraw on actual work done at the Sta. Elena Mill, are as follows:

	Cents.
Labor .....	0.03
Lubrication .....	0.01
Power .....	0.288
Tyres and Dies.....	0.129
Extra repairs.....	0.010
Total .....	0.467

"Sr. Narvaez of the Hacienda La Union in Pachuca gives a cost of 0.558 cents Mexican for reducing ore so that 80 per cent will pass a 200 mesh screen.

"At the Sta. Elena mill the ore is crushed so that 67.3 per cent passes 200 mesh.

"Against this the cost of grinding a ton of ore to approximately the same fineness by the use of stamps and tube mills will range from 65 cents to over a peso per ton, according to the size of the mill and nature of power used.

	Stamps.	Tubes.	Total.
Goldfield Consolidated give.....	0.42	0.40	0.82
Tonopah Belmont.....	0.42	0.34	0.76
Minas del Tajo, Sinaloa.....	...	...	0.96

Pachuca Mills, Approximate average, 80 to 90 cents Mex. cy.

"In these figures there is no allowance made for depreciation of mills either in Chilian mills or stamps; nor is there any allowance made for crusher power, this being the same in either case.

"When one is confronted with figures such as these it must be evident to the most casual observer that either there must be great scope for improvement in the stamps and tube mill work as at present carried out, or that through ignorance and prejudice the slow speed Chilian mill has not received the attention that apparently it deserves at the hands of engineers. On considering the matter I believe that it will be generally admitted that in the majority of cases the stamp and tube mill work as carried out in Mexico in large installations is on the whole excellent, hence we have no other excuse than to say that we as engineers have failed to credit the slow speed Chilian with the numerous advantages that, as far as I can learn, it is entitled to.

"The Chilian mills, with the exception of a few installations (the managers of which, by the way, are enthusiastic supporters of this type of mill), are as a rule a very crude type, which years ago may have given satisfaction, but now that in most cases electric power is available, and that the mechanical feeder has been perfected, and other essential adjustments, e. g., the Mantey offset and modern methods of transmission have come into general use, it does not appear reasonable that the slow speed Chilian should be condemned by those who persist in using this out-of-date type.

"The following are some of the apparent benefits to be gained by the use of Chilian mills as compared with stamps:

"Less height required. Cheaper foundations. Fewer repair parts required. Ability to withstand the roughest handling. Breakdowns and hence delays will be found less than with stamps. Over-feeding or under-feeding does not hurt the mill. In some cases absence of all screens. Absolutely unskilled labor can operate it. Attendance if mechanical feeders are used is very small. Repairs to nearly every part can be made on the ground. Minimum vibration and hence minimum depreciation of mill.

"Steel wearing parts will last from 18 months to 2 years.

"Doubtless there are other points which could be mentioned, but I think that with the above sufficient has been said to show that more attention should be paid in the future to this type of mill."

Mr. Empson has designed a mill embodying the advantages

described by Mr. Bayldon, and drawn from his own experience. He has endeavored to introduce: Improvements in the method of applying power; complete change in the form of the pan; trunnion shaft turning in a center bearing and not in the boss of the muller, where it is subjected to grit; device for allowing the mullers to lift independently over any unbreakable object; improved method of feeding ore; more discharge area which can be used either with the overflow method or with screens. The Manley offset also has been embodied. In this the axle of the wheel is set behind a diameter line to which it is parallel, so that in the turning of the mill the roller, not being true with the die, is more or less shoved over it, while at the same time it revolves. When the offset is properly proportioned to the speed at which the mill runs it is perfectly successful, but as the resistance is that of grinding, it has the objection of consumption of power and metal common to grinders, in the ratio of this action.

The estimated capacity of the proposed mill is 40 tons in 24 hours, requiring 17 to 18 hp. It will require only 5 hp. to reduce a ton of ore to practically 100 mesh.

**Cupels and Cupellation.**—In an effort to ascertain some of the factors affecting the cupellation of silver-lead buttons obtained in assaying, R. C. BENNER and M. L. HARTMANN have made a series of experiments which they have published in the *Mexican Mining Journal* of March, 1912.

The authors quote the earlier work of J. Eager and W. Welsh to show the effect of temperature, quantity of lead and quantity of silver on the cupel loss. The following tables embody these results:

*Temperature Variable; Lead and Silver Constant.  
200 mg. Silver and 10 g. Lead.*

Temp. Deg. C.	Silver Loss, per Cent.	Remarks.
700	1.02	Feathers
775	1.28	Feathers
850	1.73	No Feathers
925	3.65	No Feathers
1000	4.87	No Feathers

*Temperature 685 Deg. C.; Silver, 200 to 205 g.; Lead Variable.*

Lead, Grams.	Silver Loss, per Cent.
10	1.39
15	1.38
20	1.52
25	1.85

*Temperature 750 Deg. C.; Lead, 15 grams; Silver Variable.*

Silver Mgs.	Silver Loss, per Cent.
2	4.40
5	2.63
20	2.34
50	2.46
100	2.40
200	1.64

In the matter of specifications for cupels the authors state that no uniform conditions can be laid down on account of variations in materials and the climate of the place where the cupels are made and used. They made experiments with different mixtures to see whether suitable substitutes for boneash could be found. The experiments were made in an electrically heated muffle, and the temperatures were determined with a Le Chatelier platinum-iridium pyrometer. When thoroughly dry the cupels were heated in the muffle to about 800 degrees C. The lead button was then dropped in, and the muffle door closed until the buttons had opened and begun to oxidize. The cupels were then pulled to the front of the muffle where the temperature was about 665 degrees C. When the buttons had become quite small the cupels were pushed back into the muffle, where the temperature was from 750 to 800 degrees C.

Boneash and several other materials were tested to ascertain the silver losses, the results being summarized as follows:

*Uniform Heat Conditions; 10 grams lead, 40 mg. silver.*

Cupel.	Silver Loss, per Cent.
Morganite	1.99
Casseite	3.09
Brownite	2.89
Boneash	2.36
Portland cement	3.38
Boneash and cement (equal parts)	2.95

It was further determined that cupels made of different grades and sizes of boneash gave the same percentage loss of silver within the limits of experimental error.

**Lead.**

**Blast-Roasting of Galena.**—The early theories of the chemistry of blast-roasting of galena have been revised from time to time until some of the first proposals are now recognized as entirely erroneous. A very lengthy paper on the subject was presented to the Institution of Mining and Metallurgy (London) by Mr. C. O. BANNISTER, and published in the February *Bulletin* of that organization.

Mr. Bannister reviews at some length the various theories of the inventors, discussing their patent claims, and giving the discussion which other writers have presented on the same subject. He then gives the detailed results of a long series of experiments on the different forms of blast-roasting, and comes to certain conclusions which alone can be presented here.

1. The older theories relating to the formation and subsequent reaction of peroxides, plumbites and plumbates are wrong; when these compounds are formed it is only in minute quantity, quite insufficient to affect the final results. The former theories depending on definite reactions between calcium sulphate and lead sulphide also are wrong.

2. Later theories depending merely on the diluent effect of the various agents added are only partly true, as important chemical reactions take place in the presence of lime, limestone and magnesia.

3. The oxidation of lead sulphide takes place in three stages, (a) its partial oxidation to a mixture of sulphate and oxide; (b) a reaction between the sulphide and sulphate; (c) a reaction between sulphide and oxide.

4. In the presence of lime, limestone or magnesia, the sulphates of calcium or magnesium are formed in preference to sulphate of lead, leaving a much larger quantity of lead as oxide in the roasted material.

5. Silica and calcium sulphate act merely as diluents, taking no chemical part in the reactions until a temperature of over 1000 degrees is reached.

6. Ferric oxide in certain physical states appears to act as a catalyzing agent, lowering the temperature of oxidation of lead sulphide and increasing the amount of lead sulphate formed during the roast.

7. Silica acts at high temperatures, 1000 degrees and above, in decomposing lead sulphate and calcium sulphate, the latter being thus decomposed whether it be added as such or formed during the preliminary stages of the roast.

**International to Build Lead Refinery.**

General Manager E. P. Mathewson, of the International Smelting & Refining Company, which has a copper and lead smelter at Tooele, Utah, is authority for the statement that the company has decided to build a new lead refinery at East Chicago.

A tract of land has been purchased and plans for the plant have been completed. It is estimated that the plant should be ready for operation in about six months.

### Recent Metallurgical and Chemical Patents.

#### Iron and Steel.

**Balling Scrap Iron.**—The balling of scrap iron is a preliminary step for raising scrap to a welding temperature, and roughly welding or agglomerating it into a single mass which can then be squeezed and rolled into a homogeneous body of iron. Mr. DAVID H. LENTZ, of Chicago, Ill., has patented a mechanical device for accomplishing this object. The machine is shown in elevation and plan in Fig. 1. It consists of a fire

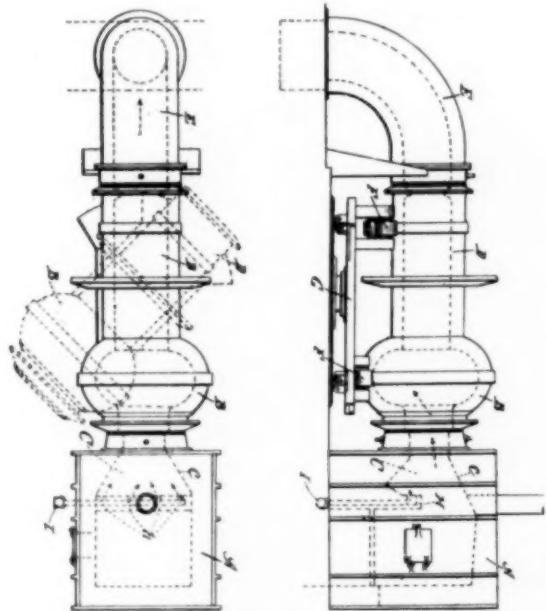


FIG. 1.—BALLING SCRAP IRON.

box *A* connecting with the balling chamber *B* and preheating chamber *D*. *B* and *D* are one member, and are adapted to be rotated on a turntable *G* into a position shown in the plan. The flue *E* is simply a down flue to carry off the gases of combustion from the furnace *A*.

In operation the furnace is first heated to a high degree of temperature in order that its refractory lining may be hot when the operation begins. The movable chamber is then swung into the position shown in the plan, and the charge of scrap iron is placed in the open end of *D*. The chamber is then swung back into operating position and revolved by any suitable means, not shown. The scrap tends to spread out over the surface of *D*, where it is subjected to preheating by the flaming gases coming from the fire box. The inclination of the preheating chamber is such that pieces of scrap are caused to move toward the balling chamber *B*; and the operation is conducted in such a manner that by the time all the pieces of scrap have reached the balling chamber they are at a welding heat. The speed of rotation may then be increased, and the mass welded into a homogeneous ball of metal ready for squeezing and rolling. Thereupon the revolving member is again swung out of operating position, the ball removed and a new charge inserted. The inventor states that he has operated such a furnace for eight hours continuously, and has welded a charge of 500 lb. of scrap in practically seven minutes. He further states that the furnace can be operated so as to weld such a charge once every ten minutes. The inventor is of the opinion that his apparatus gives results superior to those obtained by other common methods of agglomerating, in which manual labor is used, giving uncertain and irregular results. (1,102,871, Dec. 26, 1911.)

**Enriching Furnace Blast with Oxygen.**—In a patent granted to Mr. CHARLES F. JENKINS, of Washington, D. C., the inventor gives specifications for his method of using barium

oxide as a means of securing oxygen gas with which to enrich furnace blast. It is well known that barium oxide may be peroxidized and then made to yield the extra atom of oxygen taken on. This is the principle of which Mr. Jenkins takes advantage.

Fig. 2 shows a sketch of the application to a blast furnace.

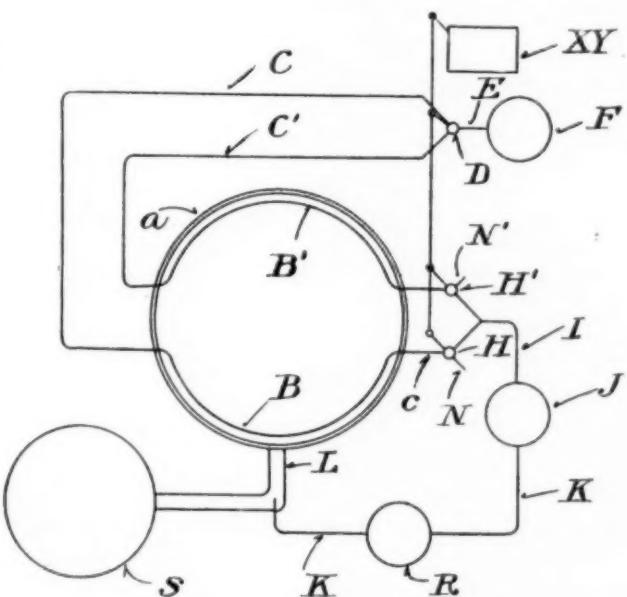


FIG. 2.—ENRICHING BLAST WITH OXYGEN.

The furnace is represented at *a*. *B* and *B'* are two receptacles for barium oxide, and are placed so that the hot gases from the furnace will surround them. In Fig. 3, we see an arrangement in which the oxide chambers are arranged in vertical twin sections in a heating chamber outside the furnace. Re-

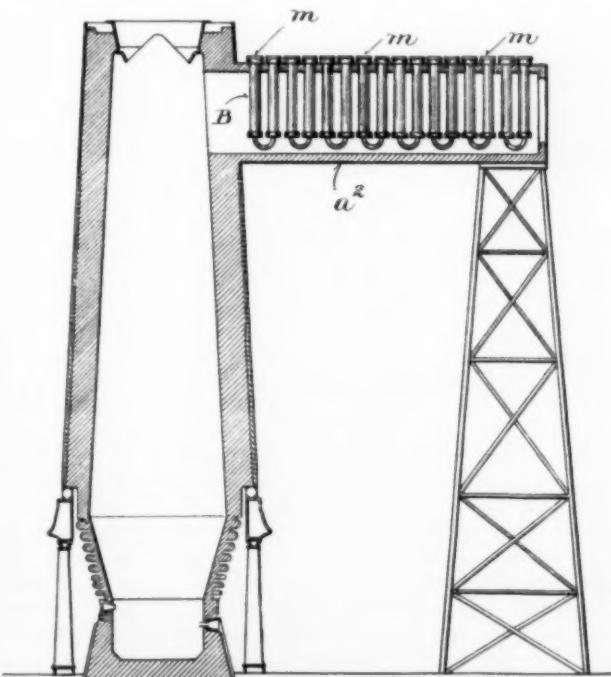


FIG. 3.—FURNACE WITH ATTACHMENT FOR ENRICHING BLAST.

ferring again to Fig. 2, the operation of the apparatus is as follows: The oxide receptacles *B* and *B'* being charged with barium oxide, the pump *J* is started which drives air through the pipe *E*, the valve *D* and the pipe *C* into the receptacle *B*, where the barium oxide takes up oxygen, forming barium

peroxide. The nitrogen passes on through pipe *G* and valve *H*, escaping from pipe *N*, unless it is desired to employ the nitrogen for some useful purpose. At the expiration of, say, 10 minutes, the three-way valves *D*, *H* and *H'* are thrown over, which closes the pipe *C* and directs the flow of air through *C'* into *B'*, where it oxidizes the barium charge and escapes through *N'*. The suction pump *J* having been started, a partial vacuum is created in the receptacle *B* and the peroxide gives up its oxygen which is drawn off through the pump *J*, the equalizing chamber *R* and the pipe *K*, to be mixed with the air blast *L*, the latter being supplied by the pump *S*. The valves are shifted every 10 minutes, and as the pumps do not stop the supply of oxygen is continuous. Any suitable means may be adopted to regulate the heat coming in contact with the oxide receptacles. (1,010,265, Nov. 28, 1911.)

**Slag Pocket for Open Hearth Furnaces.**—In the usual operation of an open hearth furnace, the accumulation of slag in the slag pockets is the cause for loss of much time in cleaning said pockets. In order to obviate the ordinary difficulties and losses attendant on this operation, Mr. THOMAS S. BLAIR, Jr., has designed a removable slag pocket which is an integral structure in itself, adapted to be attached to and detached from an open-hearth furnace. In Fig. 4 is seen a

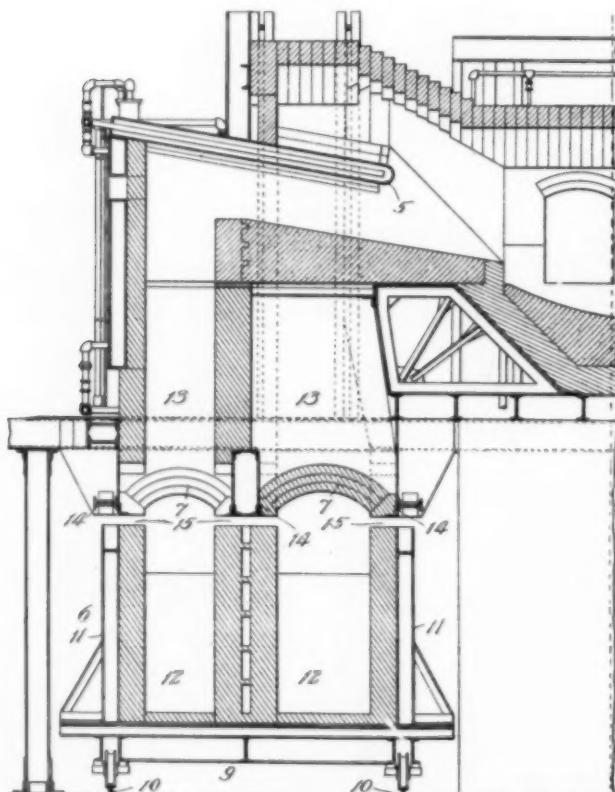


FIG. 4.—MOVABLE SLAG POCKET FOR OPEN-HEARTH FURNACE.

drawing of one end of a furnace provided with the patented slag pocket.

The slag pocket 6 complete in itself, separate from the remainder of the furn. except the slag pocket arches 7 and their means of suppo. The movable pocket consists of a wheeled carriage 9, having metal posts 11 suitably braced to provide a framework for the brick walls forming the pockets 12. The dimensions of the structure are such as to adapt it to fit under the down-takes, registering its pockets therewith. On placing a slag pocket in position there are narrow spaces at the top 15 and at the end, not shown, which are temporarily filled with brick, thus forming a tight joint between the pocket and the furnace. When it is desired to remove the pocket the bricks are knocked in and the pocket

is wheeled out of position and another moved in. Thus only a few hours' time is required to accomplish what in ordinary practice would require several days. (1,012,236, Dec. 19, 1911.)

#### Miscellaneous.

**Magnetic Removal of Dust from Roaster Gases.**—In roasting ores containing sulphide of iron the material is largely converted into magnetic forms of iron. Dust particles carried off by the roaster gases are thus magnetic and may be removed by the application of a magnet. This is the basis of a patent recently granted to UTLEY WEDGE, of Ardmore, Pa., for means of purifying roaster gases by eliminating fine solid particles which may be detrimental to the subsequent application of the gas. The customary high temperature of the furnace gases is not high enough to destroy the effectiveness of the magnets, but on the contrary the efficiency of a steel magnet may be increased thereby. (1,012,488, Dec. 19, 1911.)

#### Fused Silica Ware, its Manufacture, Properties and Uses in the Chemical Industries\*

BY A. E. MARSHALL.

The production of fused silica apparatus in sizes suitable for manufacturing operations was made possible by the Thermal Syndicate's electric-furnace process, which was patented in 1904. Prior to that date small transparent articles had been made by various experimental workers from rock crystal.

The method generally adopted was to build up the article from small pieces of crystal in an oxyhydrogen blow-pipe flame. The size of these vessels was, of course, limited, but despite this and their necessarily high cost, they were used for purposes where the peculiar properties of the ware rendered it more serviceable than other materials.

The electric-furnace process of fusing pure silica is, of course, applicable to the manufacture of large and small apparatus, the maximum sizes being pipes of 12 in. diameter and 30 in. long, basins 20 in. diameter, and retorts of 75 litres capacity. The articles made by this process are not transparent like the rock crystal product, as the mass does not become perfectly fluid during fusion, but only attains a plastic consistency with the result that the small air bubbles disseminated through it, are unable to rise to the surface.

The small ware intended for laboratory purposes is subjected to a special process which increases the transparency and gives the material a highly glazed surface. The glaze is of the same composition as the ware, that is to say, it is pure silica and is not an added alkaline glaze. The application of the glazing process is at present limited to ware not exceeding 6 in. in diameter.

Fused silica possesses properties which render it of great value in the chemical allied industries, the principal one being its great resistance to heat and to acids. It is highly refractory, the melting point (which is not well defined) being between 1700 deg. C. and 1800 deg. C. The material softens appreciably at lower temperatures (about 1500 deg. C.), but does not readily lose its shape.

There is one point in connection with fused silica that is sometimes overlooked and that is its tendency to devitrify or to pass from the vitreous to the crystalline state, if it is continuously heated above 1200 deg. C. This, of course, does not prevent the use of the material for intermittent work at temperatures considerably greater than 1200 deg. C.

The heat resisting properties of the ware received an unsolicited testimonial when the British section of the Brussels Exhibition was destroyed by fire in August, 1910. The metal fittings of the stand were fused into a shapeless mass, but the various pieces of fused silica remained intact despite the enormous temperature attained. The certificate received from the

\*A paper read before the New York Section of the American Chemical Society on March 8.

Works Department of the British Commission reads as follows: "Showcase completely destroyed, but the contents have resisted the fire. The damage to the exhibits has been caused only by the structure and iron girders falling across them."

The ware is unaffected by acids, with the exception of hydrofluoric and phosphoric acids. In the latter case the action only becomes pronounced at 400 deg. C., so that for practical purposes such as the concentration of phosphoric acid, it can be considered as being proof against this acid.

The electrical insulating value of fused silica is high, and the resistivity decreases very slowly with a rise in temperature. This suggests that the ware may be utilized for insulating purposes when the required shapes can be produced.

There are various types of electrical heaters or radiators in use in England and in Continental Europe, consisting of a heating element of non-oxidizable wire enclosed in thin walled tubes of fused silica. The tubes allow the glow of the spiral to be seen, and a very pleasing effect is produced.

In the chemical industry, fused silica has found extensive applications, particularly in the construction of sulphuric, nitric, and hydrochloric acid plants. Certain operations such as the concentration or distillation of sulphuric acid have always presented difficulties to the manufacturer by reason of the failure of the material from which the plant has been built. In some cases, costly vessels of platinum or platinum-iridium have been used to overcome the difficulties.

Fused silica has been adopted with great success for these two operations, some forty basin-type concentrators being in daily use in England alone. The silica basins are invariably exposed direct to the fire gases so that low fuel consumption and high outputs are obtainable.

There are, of course, many special applications in the sulphuric acid industry besides concentrators, but these will readily suggest themselves to the acid manufacturer.

In the case of nitric and hydrochloric acids, fused silica pipes have been extensively applied in place of earthenware. The pipes are made in the usual shapes with either socket or tapered joints. Considerable care is necessary in making the joints, owing to ordinary jointing materials and cements having coefficients of expansion considerably greater than silica. The best form of joint is one relying on asbestos for the main portion of the joint, the outside only being finished with cement.

Italian asbestos must always be used, as the Canadian variety swells considerably when brought in contact with strong acids.

A large variety of special shapes of apparatus, such as coils, tanks, retorts, crucibles, etc., are made for various branches of the chemical industry where either resistance to acids or heat, or a combination of these properties is desired.

It is interesting to note that ware of an ornamental character, such as finger bowls, fruit plates, etc., having a highly lustrous surface can be made by the same process as the chemical ware. The lustre is due to the reflection of light from the surfaces of highly distended air bubbles, and it is therefore untarnishable.



APPARATUS MADE FROM FUSED SILICA.

### The Use of Lactic Acid for Cleansing Filtering Mediums.

The new cyanide plant at the Ajax mine, in Victor, Colorado, has attracted general attention as the pioneer plant of the Clancy cyanamide process.

However, another important novelty has also been introduced there quietly into commercial practice in connection with the filtration plant, and it should be stated at once that it has resulted in complete commercial success. This is the use of lactic acid instead of hydrochloric acid for cleansing the filters.

In the past the general practice when calcium compound deposits impeded the efficiency of filtering mediums has been to dissolve them out by a dilute solution of hydrochloric acid. However, the use of hydrochloric acid has some disadvantages. The fibres of the filtering medium are rendered hard and stiff, and are completely destroyed in time. Troubles in handling the acid, high transportation charges, freight, dues on return of empties, breakages, etc., also the fact that the commercial acid of 20 degrees Beaume contains less than a third of true hydrochloric acid, make the final cost excessive. Nor has any practical method been brought forward whereby this acid, after being used, can be regenerated economically. Finally, sulphuric acid, which forms calcium sulphate on the filtering mediums, and in time causes them to become useless, is always present in commercial hydrochloric acid.

The use of lactic acid in the operation of cleansing filtering mediums is a distinct and important advance for this purpose, entirely eliminating as it does the imperfections and disadvantages just enumerated.

The incrustations of calcium compounds are not only dissolved by lactic acid as efficiently as by hydrochloric acid, but the fibres of the filtering medium, instead of being rendered hard and stiff, and finally destroyed, as is the case with hydrochloric acid, are rendered soft and pliable, therefore more efficient and of longer endurance when lactic acid is used.

Lactic acid being non-volatile and non-corrosive, is safely handled without the precautions that are necessary with hydrochloric acid. Lactic acid is transported in barrels, etc., at ordinary freight rates, instead of the high rate charged for hydrochloric acid. Lactic acid costs at the factory slightly more than hydrochloric acid, but this is more than offset by lower transportation charges and no return freight on empty acid carboys.

Lactic acid contains no sulphuric acid, therefore no incrustation of calcium sulphate is formed in the filtering mediums, as is the case with hydrochloric acid. As the efficiency of the filtering mediums is always maintained high and their life is lengthened, the cost of filtration is also reduced.

Finally, lactic acid can be regenerated for re-use from calcium lactate economically and practically. This means the elimination of delays in the delivery of fresh acid, and a partial elimination of the cost of fresh acid, since when once the bath of lactic acid has been made the only fresh acid required is for the replacement of the small amount which is lost through mechanical losses.

A 10 per cent solution of lactic acid, at a temperature of 150° Fahr., gives excellent results, as a solvent for calcium compounds from filtering mediums.

The regeneration of the lactic acid is accomplished in the following simple and efficient manner:

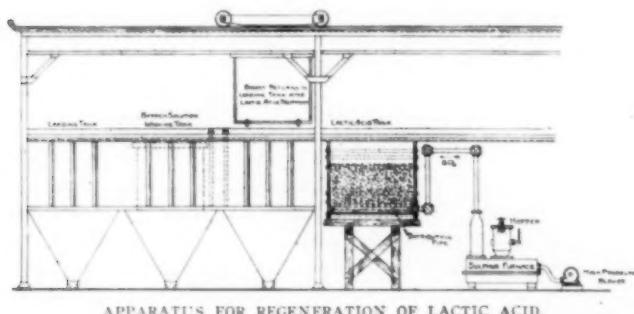
The mechanical devices required to effect the regeneration are simple in construction and operation, as is seen from the accompanying sketch in which the apparatus is shown in conjunction with a Moore filter system in cyaniding.

The apparatus required consists of a sulphur-burning furnace, an air blower, and the necessary piping, the latter being fitted up so as to connect the furnace with the perforated pipe placed at the bottom of the acid treatment tank.

The regeneration is accomplished as follows: The furnace is charged with ordinary sulphur, the latter ignited, and the

sulphur fumes forced through the calcium lactate solution, by means of the air blower, until all of the calcium has been precipitated, while the lactic acid is thereby regenerated, after which the sulphur furnace is extinguished and the operation stops.

The calcium precipitate is usually allowed to accumulate in the acid tank until sufficient has been accumulated to form a



APPARATUS FOR REGENERATION OF LACTIC ACID.

good cake on a basket of leaves when it is extracted, washed and dumped.

In practice it is found that three pounds of lactic acid are regenerated with a consumption of less than one and one-quarter pounds of ordinary sulphur.

Thus by substituting lactic acid for hydrochloric acid as a solvent for calcium compounds from filtering mediums, and by regenerating the lactic acid by means of sulphur, the cost of maintaining and cleansing the filtering mediums is greatly decreased and the efficiency of the same can be maintained at their maximum. Further, the costly and dangerous transport of hydrochloric acid is replaced by the cheap and simple one of sulphur in its place.

At the Ajax mine it has been found very convenient to remove a single leaf of the filter whenever this becomes necessary and give it a simple scrubbing with a 1 per cent solution of lactic acid at ordinary temperature, and then replace the leaf again in the basket. In view of the small amount of lactic acid used and the low freight rates to Victor, regeneration of the lactic acid is not practised at the Ajax mine.

The question whether regeneration is advisable or not depends on local conditions. In places remote from centers of supply regenerations of the lactic acid would certainly prove very convenient and economical.

The patents for this process are owned by the Moore Filter Company of New York City.

### Double Reflecting Galvanometer for Taking Recalescence Points.

By A. SCHRAMM, E. E.

The determination of recalescence points—now so well known through the more and more extended use of metallographic methods—is generally based upon the principle that chemical and physical changes within a substance are in general accompanied by an evolution or absorption of heat.

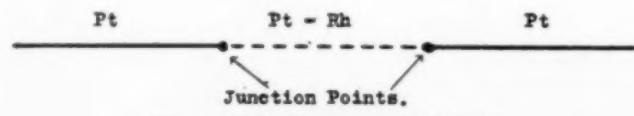


FIG. 1.—DIFFERENTIAL THERMOCOUPLE.

In this article the differential method will be considered only, and after a short discussion of the same, a detailed description of the double reflecting galvanometer will follow.

The principle of this method is to measure the temperature differences between a neutral body and the metal. This neutral body must consist of a material which is not subjected to

internal changes when heated or cooled. Platinum, porcelain or, when the temperatures are very low, even nickel, etc., serve this purpose best. It is also desirable that the neutral body has, as near as possible, the same emissivity and heat capacity as the steel to be tested. The temperature differences are measured by means of a differential thermo-couple, which consists of two platinum wires and one platinum-rhodium wire welded together, as shown in Fig. 1. When the two junction points of this element are at the same temperature, they develop the same e.m.f., but as they are connected in opposition, no flow of current can be produced.

The double reflecting galvanometer (Fig. 2), consists of

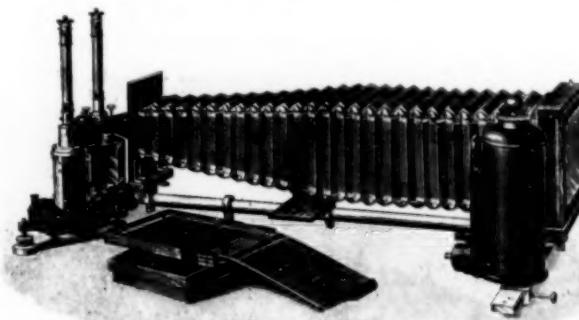


FIG. 2.—DOUBLE REFLECTING GALVANOMETER.

three distinct parts: the two galvanometers, the source of light, and a photographic camera for drawing the curves. The arrangement of these three parts is shown in Fig. 3.

A ray of light emanating from the source of light *A* falls through the lenses *C* and the diaphragms *B* on the mirror *D<sub>1</sub>* of the first galvanometer. After striking this mirror, its horizontal movement is changed into a vertical one by the reversing prism *E*. From there the light is reflected on the mirror *D<sub>2</sub>* of the second galvanometer, where its movement is again transformed into a horizontal one. Both movements result in a curve on the focusing screen *F* of the camera *H*.

Fig. 2 represents the complete outfit. The apparatus is pro-

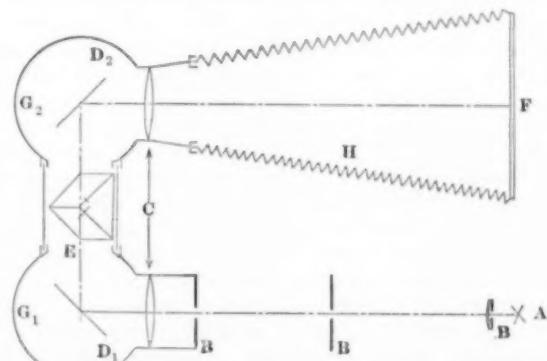


FIG. 3.—ARRANGEMENT OF DOUBLE GALVANOMETER EQUIPMENT.

vided at the lantern with two diaphragms, a larger one for observation, and a smaller one for taking curves photographically. The lenses, diaphragms, and the light tight mountings of the reversing prism prevent side lights to penetrate into the camera. It is, therefore, by no means necessary to put this machine into a dark room when taking the curves with photographic plates. When removing the plate holders or adjusting the camera as well as the lantern, jarring can not be transmitted over the light tight connection between the galvanometer *G<sub>2</sub>* and the camera, owing to the absence of any mechanical connections between these parts.

When taking curves, the neutral body and the piece of iron to be tested, have to be placed into a suitable furnace so near each other that they always have the same temperature. One

junction point of the differential thermo-couple is placed into the neutral body, and the other one into the test piece. The two free ends of this element are then connected with galvanometer  $G_1$ . By means of a regular thermo-couple the temperature of the test piece is determined and measured by galvanometer  $G_2$ .

Fig. 4 shows a curve, which was taken with a double reflecting galvanometer of the kind described above. The abscissæ

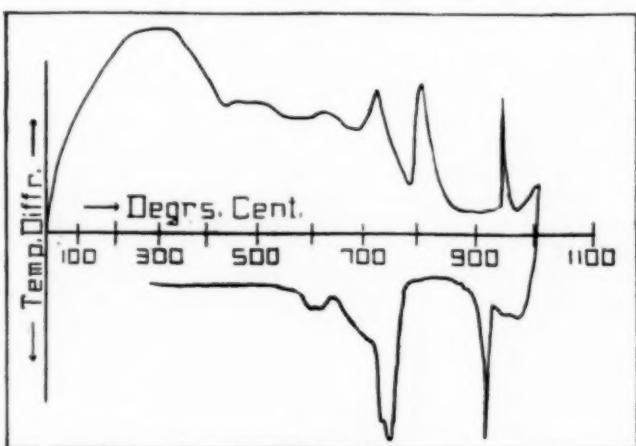


FIG. 4.—CURVE TAKEN WITH DOUBLE REFLECTING GALVANOMETER.

represent the temperature differences between the neutral body and the test piece, and the ordinates the temperature of the test piece itself. As the emissivities of both bodies were not alike, the curve does not follow the zero line, especially at the lower temperatures; nevertheless the curve shows the recalcenece points very clearly, as can be noted on the one at 950 deg. C., and it can be seen also that when cooling, the recalence point occurred nearly at the same temperature, viz.: at at 920 deg. C.

This experiment was performed in 1½ hours, which is much shorter than any of the older ways. When the instruments are set properly, the only attendance needed is the regulation of the furnace, which can be done by any unskilled workman.

#### A Very Simple Electrical Method for Testing Water.

"The Digby and Biggs Dionic Water Tester" is the title of a very interesting pamphlet by Mr. S. Evershed (of Evershed & Vignoles, Ltd., Chiswick, W., London), sent to us by Mr. James G. Biddle of Philadelphia, Pa. The fundamental idea is to substitute a measurement of electric conductivity for chemical analysis and to draw from the conductivity measurement conclusions on the amount of impurities in the water. It can be applied to measure hardness in water, leakage of cooling water into surface condensers, sewage pollution in rivers, minute traces of impurities in distilled water, the priming of steam boilers, etc.

It is clear that conductivity measurements do not discriminate between one kind of foreign substance in the water and another and that analysis alone can do that. But in most instances in which water testing is carried out for engineering purposes, the substance present in the water is known, having been ascertained once for all by analysis, or being inherent in the source of supply. Further tests are then not made for purposes of analysis, but to find out how much of the known substance is present in the water; and in all these cases the dionic tester gives the required answer with a rapidity and simplicity unapproached by any chemical test.

For instance, whenever it is important to keep a check on the purity of a water supply, the use of the dionic tester enables all analyses subsequent to the initial one to be dispensed with, until such time as change in the conductivity indicates an altera-

tion in the proportion of dissolved substances present in the water. All that is necessary is to have a sample of the water analysed and at the same time measure its conductivity. Thereafter it is only necessary to measure the conductivity at frequent intervals, and so long as there is no marked change in the value obtained, there is certainly no appreciable change in the purity. These routine measurements do not occupy more than a couple of minutes each, nor do they require a skilled man to make them. Hence the cost of testing is practically confined to the initial outlay upon the apparatus.

The construction of the dionic water tester of Messrs. Digby and Biggs is exceedingly simple. It is shown diagrammatically in Fig. 1 and an outside view is given in Fig. 2. G is a bent

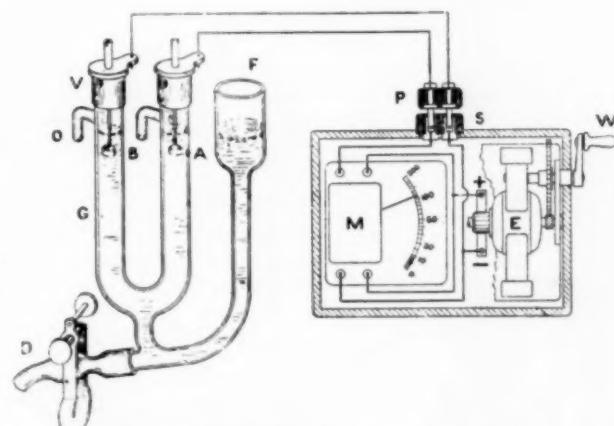


FIG. 1.—DIAGRAM OF DIONIC WATER TESTER.

glass tube to contain the water under test, and A and B are the electrodes for passing the electric current through the water. The electrodes are connected by wires to a direct-reading conductivity meter M, and a direct-current hand-driven dynamo E; so that by turning the handle W of the dynamo, a current traverses the meter and the water in the conductivity tube G. The pointer of the meter is deflected, and comes to rest at some point upon the scale which directly indicates the conductivity of the water in the tube. The test is completed as soon as the pointer has come to rest, that is to say, in two or three seconds.

The tube G is made long enough, and the electrodes are given sufficient surface, to make the electric resistance in the parts of the current-path immediately surrounding the electrodes negligibly small compared with that of the length of water in the tube. Hence, gas bubbles may accumulate on the electrodes without making any observable difference in the measured conductivity. The electrodes are short hollow cylinders of platinum so that they present a large surface, from every part of which gas bubbles are free to escape upwards.

The counter e.m.f. of polarization set up in the cell during the measurement is taken care of in advance in the calibration of the scale of the conductivity meter. Evidently, any back-electromotive force will make the conductivity of the water in the tube appear to be less than it really is; and what is done is to mark the divisions on the scale so that they represent the true conductivity of the water. To do this it is necessary to take a fair average-value for the back-electromotive force of polarization. With bright platinum electrodes in a dilute salt solution the back e. m. f. is about 2 volts, and this is the value allowed for in marking the scale.

The dynamo generates a constant pressure of 100 volts, so that any variations in back e. m. f. above or below two volts are negligibly small by comparison and do not produce any observable error. To maintain the necessary constancy of pressure the dynamo is fitted with the constant-speed clutch, which Mr. Evershed introduced several years ago for insulation testing purposes. The effect of this device is to keep the pressure con-

stant at 100 volts within one-quarter of a volt, when the dynamo handle is driven at any speed above that at which the clutch is adjusted to slip.

A correction for temperature variations is easily applied.

Conductivity or specific conductance is the reciprocal of resistivity or specific resistance the unit of which is the megohm. The unit of conductivity is the reciprocal of one megohm and has no special name; the scale of the dionic water tester extends from zero up to 2000 such units.

The conductivity tube is so constructed that by pouring water into the funnel F, and allowing it to overflow through the outlet pipes OO, every part of the tube is thoroughly washed out, and any products of electrolysis which may have accumulated upon the electrodes are at the same time swept away by the water. The air vents VV ensure a steady flow from each outlet without any syphoning action. A drain pipe D of pure rubber is provided at the bottom of the tube for the purpose of drawing off the contents at the conclusion of a test. Normally, the drain is closed by a pinch-cock. Water may be left stand-

the conductivity of which does not now generally exceed 2 units. Pure distilled water is essential for many purposes and, so far as I know, the conductivity test is the only simple means of ascertaining the degree of purity. By simple, I mean that an accurate test can be carried out by an unskilled person in about five minutes, and that no laboratory is needed."

There are several interesting appendices, one on the conductivity of mixtures, the second by Mr. F. E. Pollard on dissolved substances commonly found in water used for steam boilers, and the third giving a table of the conductivities of various waters.

Clearly the dionic water tester will not replace chemical water analysis in every case, but there are innumerable cases in chemical and industrial practice where after a single chemical analysis has been made the dionic tester can be used continuously for control for the purity of the water and can render any further chemical analysis superfluous until a change in the conductivity indicates directly the necessity of making a new one. In view of its extreme simplicity of construction and ease of use the dionic water tester should prove very handy in large-scale practice. It is being placed on the market in this country by Mr. James G. Biddle, 1211 Arch Street, Philadelphia, Pa.

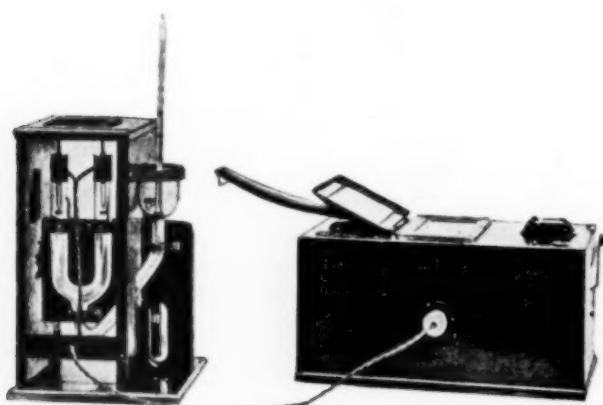


FIG. 2.—DIONIC WATER TESTER READY FOR USE.

ing in the tube for days at a time without the slightest trace of contamination from the rubber finding its way into the conductivity tube. The other materials in contact with the contents of the tube are glass and platinum.

The tube is mounted on a strong teak stand designed to afford as much protection as is consistent with accessibility.

The conductivity meter and dynamo are fixed inside a box and once for all connected up to each other and to the terminals of the socket S, which is fixed on the outside of the box. The two-pin plug P is permanently connected up to the electrodes A and B by means of a short length of well-insulated twin-flexible cable. Hence, all the user has to do in order to connect the tube to the meter is to push the plug into the socket.

The dynamo and conductivity meter do not need any attention, and, in fact, the box containing them is closed and sealed by the makers. The dial is visible through a window in the top of the box, and the handle of the dynamo is folded back into a recess when not in use.

The applications of this method to control of the hardness or softness of water, to the detection of leakage into a surface condenser, to tests of purity of distilled water, etc., are described in some detail in the pamphlet.

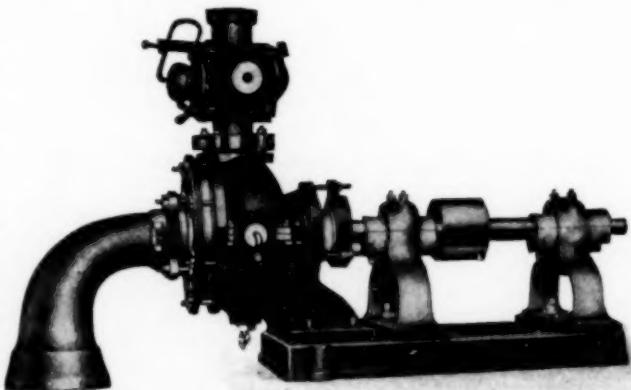
Concerning the control of purity of distilled water Mr. Ever-shed remarks: "The water tester easily detects the presence of traces of electrolytic substances in distilled water so small that they would entirely escape the analyst. One of the discoveries I made with Messrs. Digby and Biggs' apparatus was that specially distilled water, sold by wholesale chemists at four shillings the gallon as being almost the purest water obtainable, was hardly any purer than ordinary distilled water purchaseable at any chemist at a few pence per gallon. The discovery led to a considerable improvement in the specially distilled water,

### A New Acid-Proof Pump.

BY PERCY C. KINGSBURY.

The centrifugal pumps made of chemical stoneware by the Didier-March Company, of 30 Church Street, New York City, are so well known in the chemical and metallurgical industries that a short description of their latest improved model will be of interest.

One great advantage of the new design is the ease with which the interior of the pump may be examined. By loosening a few nuts the cover forming the front face of the body may be removed, exposing the impeller and allowing a thorough



STONEWARE PUMP.

inspection of the working parts. The impeller is so designed as to be readily interchangeable in case of damage by the introduction of foreign matter into the pump. The general dimensions of the pump have been retained so that the replacing of an old style pump by the latest pattern entails no disturbance of existing pipe lines. In changing this standard pattern the opportunity has been taken to make a number of minor mechanical improvements in the design, the depth of the stuffing box having been increased and the inlet of the pump made considerably larger, thus reducing the friction in the pipe line on the suction side.

All the advantages of the old-style pump are retained. Nothing but acid-proof chemical stoneware comes in contact with the liquid. The stoneware parts are accurately ground and balanced, making the action of the pump very smooth and quiet. The cast-iron casing into which the pump is grouted

effectively prevents any possibility of damage from external causes or from internal pressure. The stoneware elbow shown in the illustration is supplied with the pump and is used for connecting to a foot valve when required. This foot valve can also be obtained made entirely from chemical stoneware. The three-way cock serves for priming the pump and is also supplied.

The greatest care is given to the manufacture of the stoneware impeller which in the smallest size pump runs at 1500 r.p.m. All impellers are tested at 50 per cent of their rated speed before the pumps are assembled. A series of experiments have been carried out on these pumps by Professor Schulze-Pillot at the German works of the Didier-March Company to establish by actual operation under working conditions their efficiencies and capacities under various heads and at various speeds. Reliable information in regard to the use of the pumps for different purposes is therefore available. An experimental station will shortly be ready at the factory of the Didier-March Company, Keasbey near Perth Amboy, N. J., where parties interested may see these pumps in operation.

In handling acids and other corrosive liquids the continual breaking down of metallic pumps with the constant repair and lost time has been a serious problem, frequently involving danger to workmen and property. An incorrodible pump of practical design, developed by years of experience, should prove a satisfactory solution of many such difficulties and has, in fact, for many purposes proved to be absolutely indispensable.

### Steam-Jacketed Kettles, Mixers, and Vacuum Pans.

The Sowers Manufacturing Company of 1302 Niagara Street, Buffalo, N. Y., have just issued a profusely illustrated catalog (Catalog V, section 1) on the steam-jacketed kettles, mixers and vacuum pans of their make.

Since the H. W. Dopp Company was taken over by the Sowers Mfg. Co. the well-known Dopp seamless steam-jacketed kettle is now being manufactured by the latter company, and in view of their greater manufacturing facilities some interesting advances in its construction have been made.

The Dopp kettle is seamless, so that all joints are done away with. The inner and outer shells are absolutely cast in one piece, thus avoiding joints and rivets, and the inner and outer shells are tied together by a system of oval stay-bolts cast at the same time as the shells. This construction permits the use of thin shells, which means a highly efficient kettle through perfect circulation and quick radiation.

By the system of molding used by this company sand does not come in contact with the inner and outer surfaces of Dopp kettles. This insures a perfectly clean and smooth casting, without the necessity of grinding the surface for a smooth finish, which destroys the hard skin valuable to the life of a kettle. The catalog gives illustrations and data on different types and sizes of these steam-jacketed kettles, and especially on the different types of standard outlets used in connection with them.

The second part of the catalog gives a series of illustrations of steam-jacketed agitators and mixers with various arrangements of the paddles, scrapers, screws, etc., to provide the specific mixing operation required in any case. These mixers being steam-jacketed, also permit quick and effective cooling, when necessary, by running cooling water through the jacket.

The concluding part of the catalog is devoted to seamless jacketed vacuum pans.

The basis of these vacuum pans is again the Dopp seamless steam-jacketed kettle used as the lower half. The dome being practically the same shape as the kettle, makes a very symmetrical and strongly proportioned piece of apparatus. The pans are cast by the same special process which gives to Dopp kettles their smoothness and strength. This makes them particularly well adapted for use where a perfect surface and cleanliness are desirable and necessary.

These vacuum pans are made in 15 standard sizes, ranging in capacity from 10 to 1,300 gallons. Various types are shown in the illustrations in the catalog, some of them provided with agitating devices, also steam-jacketed autoclaves or digestors, for use in the manufacture of materials which require cooking under pressure.

The catalog, which is concluded by data on and illustrations of vacuum pumps, is well worth the attention of the many users of steam-jacketed apparatus in the chemical industries.

### A New Protective Coating for Iron and Steel.

Since the introduction of sherardizing into commercial practice a few years ago, the Lohmannizing process to be described in this article is probably the most interesting and novel development in the endeavor to find a new solution of the ever-burning problem of providing a really protective coating for iron and steel.

If we leave aside protective paints, protection has generally been sought in the past by means of zinc coating. There are three zinc plating processes at present in extensive commercial use: the old "hot galvanizing" process, the electrolytic zinc plating or "cold galvanizing" process, and the more recent "sherardizing" process. Concise descriptions of these processes, as well as critical comparisons between them, may be found in an extensive article by Mr. Sherard-Cowper-Coles in our Vol. VI, p. 189 (May, 1908).

The process of Lohmannizing, called after its inventor, Mr. Herman J. Lohmann, differs distinctly from the processes just mentioned in so far as it is by no means restricted to zinc coatings, but may be used for coatings of zinc, lead, and tin in varying preparations to suit the requirements of each case.

The patents for the process are owned by the Lohmann Company of 50 Church Street, New York City, which is introducing the process into commercial use by granting licenses.

Licenses have so far been granted to the Brier Hill Steel Company of Youngstown, Ohio, and to the DeForest Sheet & Tin Plate Company of Niles, Ohio.

The Erie Railroad Company has also used the process quite extensively during the past two years for plating whatever was suitable around their locomotives, and has achieved excellent success.

The Lohmannizing process is very similar to the old hot galvanizing process. The similarity is indeed so close that workmen who are accustomed to work on the hot galvanizing process can be put immediately on the Lohmannizing process without even knowing it, since the chief difference is that instead of the muriatic acid bath used in hot galvanizing just before the article enters the spelter bath, the so-called Lohmann bath is employed as will now be described. The following description is based partly on an account given out by the Brier Hill Steel Company and partly on the patent specifications.

The process begins as usually with pickling the iron and steel in a bath of sulphuric acid. This pickling treatment removes rust, dirt and scale in the well-known manner from the surfaces which are to be coated.

Now the article is dipped into the Lohmann bath, which, "being composed of an acid and an amalgamated salt, further cleanses the pores and cavities, and deposits metallic salt upon the entire surface, penetrating into the most minute pores and cavities." According to the patent specification the Lohmann bath is a solution of hydrochloric acid, bichloride of mercury, and sal ammoniac. The essential new feature is the bichloride of mercury.

Now follows drying and then immersion in the molten protective alloy which is at a temperature of 950 to 1000 deg. F. "An amalgam or chemical union is thus formed between the amalgamating salt and the protective alloy." In contact with the molten alloy the temperature of the immersed iron or steel article gradually rises. When it reaches a temperature of 500

deg. the mercury volatilizes, is released, and rapidly comes to and passes from the surface of the alloy.

While this occurs the complete surface of the treated metal, especially its pores and cavities, are left free and open for the protective alloy to fill them up and enter into a perfect union with the iron and steel surfaces, there being a complete freedom from oxidizing influences.

In this way it is possible to get a perfectly satisfactory lead plating as is not possible with other methods.

The chief points of the process are that the junction be-

it is also possible to use the Lohmann process for zinc plating or for applying a protective alloy of zinc, lead, and tin in any proportions.

For instance, a coating of 100 zinc, 4 lead, and 1 tin is useful as a substitute for galvanized sheets where it is desired to make the sheets more resistive and pliable.

A coating of 10 lead, 1 zinc, and 1 tin is useful for articles where extreme pliability is not needed and where a bright appearance is desired in the finished article. Of course, such a coating is not as resistant as straight lead.



GALVANIZED SHEET.



LOHMANNIZED SHEET.

tween the iron and steel and the protective alloy is not only maintained chemically pure and free from oxides, but is also mechanically intimate.

That this result is really obtained is confirmed by the microphotographs of the cross-section of a hot galvanized sheet (Fig. 1) and of a Lohmannized sheet (Fig. 2), both being magnified 65 times. For these microphotographs we are obliged to the Brier Hill Steel Company.

Fig. 1 indicates that the coating on a hot galvanized sheet resembles in its nature a coat of paint. The spelter coat, when thick as it should be, is liable to crack. When thin it is porous. In either case the iron sheet would be unprotected. Also minute particles of the acid used in pickling or cleaning the metal are liable to be carried over on the sheet, through the spelter pot, and finally be included between the sheet and the coating. The acid in time causes oxidization, the coating is bulged out from the sheets and, at least, it crumbles and chalks, leaving only the bare sheet.

Fig. 2 shows, in the case of Lohmannized sheet, the permeation of the sheet surface by the alloy. This continuous alloying zone between the outer coating and the underlying iron and steel cannot be ruptured in such a way as to leave the sheet exposed without fracturing the sheet itself, so that the sheet is always protected. Also the "amalgamating salt" of the Lohmann treatment cleans the sheet absolutely and itself evaporates wholly as the sheet receives the alloy, thus insuring the absolute elimination of all oxidizing agents such as would naturally cause crumbling and destruction of the spelter layer.

The underlying idea of Mr. Lohmann's work was to get away from pure zinc plating. As described it is possible to obtain a pure lead plating by the Lohmann process. To our chemical readers who daily use lead-lined apparatus the applications of this process for such purposes are evident. But

It will be easily seen that according to the requirements of each case the composition of the coating can be varied. It should be added that the chemicals used in the process are quite inexpensive, while the use of lead means, of course, a considerable saving.

The Lohmannizing process certainly desires careful watching in its introduction into commercial practice.

#### Non-Ferrous Metal Market.

Copper and lead have been almost without feature since our last report. Tin has fluctuated somewhat, and the market has been affected by the English coal strike which, it was thought, might seriously affect shipments and cause a shortage. Spelter has been rising steadily, although the market has shown wide variations in prices.

**Copper.**—This market has been without feature, and sales have been small owing to the high prices asked. The principal sellers have maintained their prices in spite of the dull market, and feel hopeful that sales will increase at the advance asked. The latest reports quote Lake at  $14\frac{1}{2}$  to  $14\frac{3}{4}$  cents per pound, and electrolytic at 14.40 to 14.45 cents.

**Lead.**—The leading seller has continued to meet all demands at the uniform price of 4 cents, New York, and 3.95 cents, St. Louis.

**Tin.**—The domestic market has declined in sympathy with the London market, which has been depressed on account of the English coal strike. Domestic buyers have kept out of the market largely and are awaiting developments. March tin was quoted at about  $41\frac{7}{8}$  cents.

**Spelter.**—Prices for this metal have risen steadily during the past month. The smelters seem to be well sold out and spot metal commands a premium. The market has been very

irregular and prices have varied widely. The last quotations are 6.65 to 6.95 cents, New York, and 6.50 to 6.80 cents, St. Louis.

**Other Metals.**—The market for aluminum is fairly good and the March price was about 19½ to 19¾ cents, New York. In antimony there has been little change and small volume of business; the prices vary from 6¾ to 7¾ cents for different brands. The market for quicksilver is a little firmer and prices unchanged at \$46 per flask of 75 lb. at New York and \$45 at San Francisco.

### Personal

**Messrs. Charles H. Repath and A. G. McGregor** have formed a partnership for the conduct of an engineering business, with a specialty of constructing metallurgical plants. They have been associated together in the construction of some of the largest plants in the West, and now have charge of other important works. Their office is at Douglas, Arizona.

**Mr. H. Kenyon Burch** is to have charge of the remodeling of the Old Dominion concentrator, at Globe, Arizona. He is now at Globe preparing plans and specifications.

**Mr. Hiram W. Hixon** has left Philadelphia to take charge of the smelting works of the Tezuitlan Copper Company, Tezuitlan, Puebla, Mexico.

**Smith, Emery & Co.**, Los Angeles, are installing a complete ore-testing plant in connection with the chemical laboratory at Los Angeles, Cal.

**Mr. F. W. Draper** is to have charge of the design and construction of a new copper smelter at the Kalata mines of the Verk Isetz Corporation, of Ekaterinburg, Russia, and will also remodel the existing plant at the Pushminsky mines.

**Mr. Cyrus Robinson**, of New York, has charge of the installation of a new cyanide plant at Fresnillo, Zacatecas, Mexico.

**Mr. A. J. Hoskin**, formerly professor of mining at the Colorado School of Mines, has been appointed Western representative of *Mines and Minerals*, with offices in Denver, Colo.

**Mr. George J. Hough**, of Salida, Colo., has sold his assay office at that place, and will again engage in professional work as a metallurgist.

**Mr. A. H. Brown**, formerly metallurgist at the Nickel Plate property, Hedley, B. C., has accepted a position with the Hudson Bay Mines, Ltd., Cobalt, Canada.

### Notes

**Connersville Blowers at the Granby Smelter.**—In the article on the Granby smelter in our last issue there is an error on the bottom of page 151 in so far as mention is made of only one 30,000-cu. ft. Connorsville blower; but the picture shows four of these machines in one room, each driven by two electric motors. The Connorsville Blower Co. considers this an ideal method of driving rotary blowers, because each impeller is driven independently.

**The Goldfield Consolidated Mines Company**, Goldfield, Nev., has issued its final statement of November, 1911, operations. The 100-stamp mill treated slightly over 954 tons per day, which is less than the high record for October. The cause lay in the time lost in relining tube mills and repairing Chileans. The average value of ore was \$26.77 per ton, and the extraction netted \$25.28 per ton, or 94.45%. The total cost of mining, milling, office and general expense was \$7.49 per ton.

**Ferrotungsten** is produced electrically, according to the *Mining Journal*, by the Ampere Company, of Berlin. Scheelite is smelted directly with iron sulphide and carbon in an electric furnace, the iron sulphide acting as a flux. Lime also is added to flux the silica in the scheelite. The reaction given is as follows:



The lime silicate slag carries the calcium sulphide formed according to the reaction.

**The Serpek process of nitrogen fixation** consists in heating bauxite with coal at a temperature of 1700 to 1800 deg. C. From 15 to 18 per cent nitrogen is absorbed as aluminium nitride. The nitrogen-carrying bauxite is then treated with caustic soda, whereupon ammonia is evolved and caught. The profit from the ammonia reduces the cost of alumina and aluminium.

**The production of tungsten ore** in the United States was not recorded previous to 1900; before that time it was insignificant, and it is only since 1906 that the data have been kept in a manner to make comparisons possible. The output since that year, expressed in short tons of 60% tungstic acid content, is as follows: In 1906, 928 tons; 1907, 1640 tons; 1908, 671 tons; 1909, 1619 tons; 1910, 1821 tons.

**The output of tin-plate mills** in the United States has shown a rapid increase in the past two decades. In 1892 the total output of American mills was 18,803 tons; in 1900 it had risen to 302,665 tons, and in 1910 it reached 725,000 tons, making the United States the world's chief producer of tin plate. Statistics regarding imported plate and re-exported containers show that practically all the plate imported in 1911 was again sent out of the country as manufactured cans, etc.

**The approximate value of silver ore and concentrates** shipped from Cobalt, Canada, during the last six months of 1911 was \$6,247,077. There are now 16 concentrating mills in active operation in the Cobalt district.

**The heat of formation of titanium dioxide** has been determined by W. G. Mixter to be about 218,000°. His best results gave 218,400°, and the mean of seven experiments was 218,500°. Mr. Mixter was led to repeat his previous work, in which he obtained a figure of 215,600, on account of the work of German investigators who gave as the average of their experiments the figure 97,772. The full details of Mr. Mixter's work are published in the *American Journal of Science*, for January, 1912.

**The slow-speed Chilean mill** is essentially a fine grinder. Where ore is to be crushed to ¼ in. in stamps and then ground to 30 or 40 mesh in tube mills, the slow speed Chilean cannot compete, but as a fine grinder it is regarded by many engineers as having a high mechanical efficiency.

**Scaling Amalgamated Plates.**—The following procedure is adopted in South Africa for removing the hard scale of amalgam which forms on copper plates after they have been in service for a long time. The plate is heated over a wood fire to drive off all the mercury, the heat being just sufficient to char saw-dust thrown on the plate. It is then removed, and while fairly warm is painted with strong hydrochloric acid. After standing over night the plate is painted with a mixture of equal quantities of saturated solutions of saltpetre and sal ammoniac, and heated again over a bright wood fire. At this stage the scale rises, and after removal of the plate from the fire may be brushed off. The treatment of the scale is similar to that of zinc-gold slime. Usually the first scaling will remove most of the gold, but a second operation will give a slight additional recovery.

**Transvaal Gold Production.**—The number of companies reporting to the Transvaal Chamber of Mines in November, 1911, was 68. The total tonnage milled during that period was 2,138,594 tons of ore; 10,162 stamps were in operation with an average duty of 8.01 tons per 24 hours. Tube mills in commission numbered 243. The yield for the month was 719,729 fine ounces gold. The record for the first eleven months of 1911 eclipses that of the entire year 1910.

**The Leadville Mines Pumping Company** has received favorable responses from many of the owners of Fryer hill properties, and has prepared plans of operation so that when perfect co-operation is finally assured no time will be lost in getting the project under way. Carefully drawn contracts

based on fairness and justice to all concerned will soon be ready to send to various owners and lessees for their signatures.

The Ore Concentration Company, Ltd., reports that during the month of January the Elmore plant at the mines of the Sulitelma Copper Company, Norway, produced 720 tons of copper concentrate.

**Bleaching Powder.**—In a paper by R. L. Taylor, originally presented before the (British) Chemical Society (*Journal*, Vol. 97) and recently republished in the *Journal* of the Municipal School of Technology, Manchester, Vol. IV, the following conclusions are reached: The action of carbon dioxide on bleaching powder and similar substances results in the liberation of chlorine only—no hypochlorous acid. The conclusion is drawn that the action is like that of any other acid, and that carbonic acid decomposes both the chloride and the hypochlorite in the bleaching powder. It follows from this that the action of hydrochloric acid on carbonates is a reversible one. Ordinary moist air acts on solid bleaching powder, liberating at first both chlorine and hypochlorous acid, the former in much the larger amount. After a time nothing but chlorine is produced. When ordinary air is passed through a solution of bleaching powder, a mixture of hypochlorous acid and chlorine is swept out, at first in about equal amounts, but as the experiment proceeds, the former diminishes and the latter increases to about 90 per cent. The action of chlorine on alkalis, like that of iodine and bromine, is a reversible one, as stated by von Tiesenholz. If the free lime in bleaching powder is removed, this causes the reverse action to proceed and thus chlorine is liberated. This explains the action of ordinary air on bleaching powder. The reversibility of the action also explains the stimulating effect on bleaching which the addition of calcium chloride or of salt causes in a solution of bleaching powder. In the ordinary processes of bleaching the active bleaching agent is probably free chlorine, hypochlorous acid playing only a minor part.

The firm of Robt. W. Paul of New Southgate, London, England, manufacturers of electrical standard and testing instruments, has opened a branch depot and laboratory at 1 East 42d Street, New York. This branch is in charge of Mr. Cuthbert J. Brown, who for some years has been engaged in the testing department of Mr. Paul's works, and who will be responsible for the good condition and accuracy of the instruments shipped to purchasers in this country. We have received an abridged catalog containing particulars of the very large range of instruments manufactured by Mr. Paul, including a series of moving-coil instruments on the unipivot principle. These instruments are robust and free from the necessity of leveling, although they have an extremely high sensitivity and can therefore be used in place of reflecting galvanometers for many tests, as well as forming convenient resistance millivoltmeters and microammeters.

## BOOK REVIEWS.

**Sulphuric Acid and Alkali.** Vol. III: Ammonia, Soda, Alkali and Chlorine. By G. Lunge. Third edition, 6 x 9 in., 764 pages, 184 illustrations; price, \$10 net. New York: D. Van Nostrand Company.

The venerable mentor of the acid and alkali industry gives us again a painstaking and detailed revision of his third volume. He includes in it chapters on bleach and chlorates, but excludes electrolytical methods, which are to be handled by Professors Haber and Askenasy, and published later as Volume IV.

Dr. Lunge's many friends rejoice that he is still able to render such signal service to the chemical industry; his wonderful volumes have served not only as a *vade mecum* to the acid and alkali industry, but have set a standard in scientific treatises which has benefited by its example the whole field of technical literature.

**Messungen elektromotorischer Kräfte galvanischer Ketten mit wässrigen Elektrolyten.** Gesammelt und bearbeitet in Auftrage der Deutschen Bunsen Gesellschaft von R. Abegg, Fr. Auerbach und R. Luther. 16 x 24 centimeters, 213 pages; price, bound in paper, 8.40 marks. Halle a.S.: Wilhelm Knapp.

The aim of this work is to present "a tabulation like that of Landolt-Bornstein-Meyerhoffer's for physico-chemical data, which shall give in this field a complete review of all the investigations so far made, in order to facilitate the use of these data for chemical purposes." The volume contains three distinct sections: (1) A systematic, chronologically arranged list of all the literature of such measurements; (2) a systematically arranged selection of the most reliable results of such measurements; (3) tables of the present most probable values of single potentials. For carrying on the work, the authors request the co-operation of all persons making such measurements, by sending of results to Dr. Auerbach, care of the publishers, Wilhelm Knapp, Halle a. S., Germany.

**Galvanostegie.** I Theil: Über elektrolytische Metallniederschläge. Von Dr. Ing. M. Schlöffer. 17 x 24 centimeters, 257 pages, 22 illustrations. Price, in paper, 12 marks. Halle a.S.: Wilhelm Knapp.

The author was practically engaged in the Langbein-Pfanhauer Works. He intends the book to present not merely the facts of electroplating and the composition and handling of baths giving good results, but a scientific discussion of the "why" and "wherefore," as far as such explanation can be given. Writing on this plan, the author has made a very instructive and interesting book. His chapter on the influence of colloids in metal plating is a valuable contribution to the subject, and the same may be said of a 30-page chapter on the deposition of alloys. The book is worth careful study by all intelligent electroplaters.

**Metallurgy.** Vol. I: Introductory. By Herbert Lang. 6 x 9 in., 255 pages, 24 illustrations; price, \$3 net. New York: McGraw-Hill Book Company.

Mr. Lang is known as the author of a small practical book on "Matte Smelting," and he now puts into effect a plan of sixteen years' standing, viz., to discuss the fundamental principles of metallurgical processes and the essential features of metallurgical apparatus and appliances. As the author says, metallurgical literature is rich in descriptive works, but lacks a comprehensive study of "the reasons why." The lack is not so glaring at present, we may remark, as it was sixteen years ago, when the plan of this work was conceived, but there is still room for such a work if written by the right man and in the right way.

This introductory volume covers only a few topics: viz., ore crushing, sampling, roasting, smoke and fume. Volume two will include smelting processes, and volume three amalgamation and wet methods. In the chapter on ore crushing the author has very cleverly attained his object; the discussion of principles, illustrated by a few examples of apparatus, is excellent and will prove very instructive. Sampling is similarly treated, but not so fully. Roasting is voluminously explained, but not in terms quite as exact or definite as the present state of metallurgical knowledge would warrant; the *rationale* of self-supporting roasting is not treated as thoroughly as it should have been, particularly in connection with pot roasting. However, these are minor omissions in a very well-written and illuminating chapter. Smoke and fume are handled excellently, with some original ideas as to why and wherefore, however, which are not accepted everywhere as orthodox.

The style is familiar, and will appeal to the student; the English is not excellent, but is usually clear; the explanations are necessarily colored by the author's preferences, but are generally sound and in all cases put in a way stimulating to observation and thought.

It is a useful book, which has an undoubtedly welcome awaiting it in metallurgical circles.